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METALLURGIST

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IN ENGLISH TRANSLATION

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SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

ANII	Arctic Scientific Research Institute
FIAN	Physics Institute, Academy of Sciences, USSR
Geologizdat	Geological Press
GGGT	Geological-Hydrological-Geodetic Trust
GGI	State Hydrological Institute
GGK	Main Geodetic Committee
GGO	Main Geological Observatory
GGU	Main Hydrographic Administration
GITI	State Scientific-Technical Press
GITTL	State Technical and Theoretical Literature Press
GOI	State Oceanography Institute
GONTI	State United Scientific-Technical Press
Goskhimizdat	State Chemical Press
GOST	All-Union State Standard
GTTI	State Technical-Theoretical Press
IGD	Institute of Mining
IGN	Institute of Geological Sciences
IL	Foreign Literature Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Academy of Sciences USSR Press
Izd. MGU	Moscow State University Press
MGI	Moscow Geodetic Institute
MOIP	Moscow Society of Naturalists
NGRI	Petroleum Geological Prospecting Institute
NIIG	Scientific Research Institute of Hydro-Engineering
NIIGGR	Scientific Research Institute of Geophysical and Geochemical Methods of Prospecting
ONTI	United Scientific-Technical Press
OTI	Division of Technical Information
OTN	Division of Technical Science
PIN	Paleontological Institute (of Acad. Sci. USSR)
TsNIGRI	Central Scientific Research Institute of Geological Surveying
TsVTI	Central Office of Economic Information
UF	Ural Branch
VGO	All-Union Geographical Society
VGRO	All-Union Geological Prospecting Association
VNIGRI	All-Union Petroleum Scientific Research Geological Prospecting Institute
VNII	All-Union Scientific Research Institute of Instruments
VNIIGAZ	All-Union Scientific Research Institute of the Gas Industry
VSEGEI	All-Union Geological Scientific Research Institute

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us. – Publisher.

BETTER QUALITY PRODUCTION, LESS SPOILAGE

I. P. Zubrev

The sixth Five-Year Plan for the development of the national economy of the USSR provides for a considerable increase in the manufacture of iron, steel and rolled steel.

One way of raising production is by improving the quality of the product and reducing spoilage.

During 1956 in the factories of the south alone there was lost in spoilage 17,950 tons of pig-iron, 199,710 tons of steel and 173,007 tons of rolled steel.

In addition there was a further loss of 6,650 tons of rolled steel through claims from the users.

A reduction in the spoilage in the southern steelworks could in 1956 have yielded additional tens of thousands of tons of pig-iron, steel and rolled steel.

In 1956 as compared with 1955, there was a considerable increase in spoilage of pig-iron – from 0.032% to 0,108%. The amount of pig-iron falling short of the internal technical specifications applying at the factory was 2.74% of the whole of the output.

In particular, the output of reject production at the Voroshilov and Kramatorsk plants increased respectively by 71.5 and 57%.

The increased spoilage at these plants occurred mainly on account of a falling off of quality in the raw materials supplied. For example, deficiency in deliveries to the Kramatorsk plant of manganese ores of of the appropriate grade and consistency of chemical composition led to frequent overburdening and disturbance of the furnace operation.

The sinter delivered by the Krivorozhsouthern ore treatment plant to the Voroshilov works had a low mechanical strength, high humidity, and poor caking and displayed large variations in chemical composition. This also gave rise to frequent overburdening and disturbance in the operation of the furnaces.

A second, no less important cause of increased spoilage of pig-iron was the irregular technology applied to ore treatment in the ore piles, blending, ore and slag tapping program, cleaning the ladles of slag and so on.

During nine months of 1956 there was a considerable increase in spoilage of steel at the Dzerzhinsky plant (from 1.73 to 2.70%), the Makeevka plant (from 0.59 to 0.99%) and Enakievo (from 0.8 to 0.90%). There was also a high level of spoilage at the Voroshilov plant, namely 2.31%.

Of the total spoilage at the steel smelting plants, losses in the first reduction comprised 52.1%, losses through breakdown and short pouring – 33.0%, and ingot rejects 14.9%.

The high percentage of spoilage in the first reduction is explained by the low quality of the ingots. The large breakdown losses indicate frequent departures from the technical codes of practice at the plants.

The amount of reject steel at the Dzerzhinsky plant was 87% higher than at all the open-hearth furnaces of the Kuibyshev and the Magnitogorsk steel works.

The worst figures came from No. 3 open-hearth furnace where the reject figure amounted to 3.39% (39,502 tons).

The low labor and technological discipline at this plant has resulted in a high level of technological disturbances. Hence, in the third quarter of 1956 80% of the smelts were carried out with technological disturbances.

Frequent disturbances in the technology resulted in low-quality ingots, giving rise to a high level of first reduction reject (70% of all the spoilage).

The high reject level at this plant is also explained by the fact that the technology of smelting and pouring tube steel and rimmed chrome steel has not yet been completely mastered. Spoilage in the smelting of tube steel is 7.45% and of rimmed chrome steel 5.4%.

At the Makeevka plant the worst figures were yielded by the No. 1 open-hearth furnace. In the third quarter of 1956 spoilage amount to 1.26% as against 0.61% in 1955. Stoppage losses in steel pouring were increased by 83% as compared with 1955 and breakdown losses attributable to the furnace bay by 16%, under-pouring increased in the ratio of two to one and ingot rejects in the ratio 4.6 to 1, and spoilage in the first reduction in the ratio of 2.8 to 1. The increased spoilage is the result of a weak technological discipline resulting in a sharp increase in the cases of disturbed technology (1.17 disturbances per smelt as against 0.67 disturbances in 1955). During the third quarter the charging technology was disturbed in 48.9% of the smelts, the reduction technology in 17% of the smelts while in 37.3% of the smelts the quantity of slag run-off was less than the established standard. In addition, there were frequent disturbances in the metal reduction on account of prolonged opening of the tapping hole, high sulfur content, lack of ladles, slag pots, crews and so on.

In the pouring bay of this plant there is a low technological discipline and an absence of supervision over the work performance. As a result of this there are frequent burner irregularities for bottom heating of the pot in the ladles, anddelivery of the ladles with undried pots. In the stopper drying house the temperature regime was not observed and the stoppers were inserted in the ladle incompletely dry, burning out in tapping.

At the Voroshilov plant the steel spoilage was intolerably high. Spoilage in the first reduction comprised 52.3%, stoppage and under-pouring losses 37% and ingot rejects 10.7%.

The high percentage of spoilage in the first reduction was caused by the low quality of the ingot, insufficient dressing of the ingot and billet, together with the high amount of departures from the technological codes. This explains the large percentage of breakdown metal losses.

The plant management is not taking the necessary measures for removing the causes of technological disturbances. The decisions adopted at meetings and instructions put out at the plant, aimed at improving the quality of production and reducing spoilage, are not always put into effect.

During nine months of 1956 a number of plants increased the spoilage in rolling: Enakievo from 0.78 to 1.43%, Dzerzhinsky from 1.48 to 1.55% and Komintern from 2.58 to 3.66%.

There is at the same time a high reject percentage-at the Petrovsk plant, 2.14% and the Zaporozh.stal 1.89%-although these figures are lower than for 1955.

The Komintern plant shows a high reject figure for rolled steel. Of the total spoilage at this plant 84% is attributable to rolling rejects, explained by the poor adjustment of the rolls, flying shears and guides. In the third quarter alone the technical inspection service had to stop the mills twenty-five times in order to forestall mass spoilage.

At the Dzerzhinsky rolling mill 9,349 tons of rolled steel were rejected through short rolling, curling and other defects occurring as a result of operation on badly adjusted mills, careless operation by the rollers, frequent shut-downs and faults on the mills.

As a result of inadequate dressing of the slabs and billets, poor scale-cleaning of the sheet in rolling, and operating with worked-out fittings and rolls, rejects through surface defects comprised 8,745 tons.

There was also enormous spoilage through defects in the steel smelting performance and rejects attributable to the open-hearth furnaces. In the third quarter of 1956 alone the blooming mills contributed 7,134 tons additional crop ends to steel smelting through defects in the steel smelting performance.

At the blooming mill 1,150 crops were cut inaccurately, as 4-5 rolled slabs were cut simultaneously with different lengths. As a result of this, part of the sound metal went to scrap and part of the metal carrying defects was passed to the second reduction.

Serious danger is caused by the quality of rails and axle billet on account of flaking. In the course of nine montsh 109 rail and axle billet smelts were rejected on account of flaking.

It can be seen from the figures set out above that several factories carry large unutilized reserves for increasing steel and rolled steel production in the form of reduced spoilage, increased quality of production and complete elimination of deliveries of low quality and sub-specification products to the users.

PROJECTS OF THE SIXTH FIVE-YEAR PLAN

THE SOKOLOV-SARBAI ORE CONCENTRATION PLANT

N. F. Sandrigailo

Director, Sokolog-Sarbai Ore Concentration Plant

The Sokolov-Sarbai Ore Concentration Plant.

The main ore base for the iron and steel plants of the Urals is at present the Mangitnaya mountain with the Uralruda mines. These concerns are under great pressure to provide all the ever-increasing ore requirements of the blast furnaces already existing. They are not, however, in a position to satisfy the requirements of the blast-furnaces and open-hearth furnaces in construction.

The Kustanai site contains enormous ore reserves not inferior in quality to the ores of the Krivorozh basin and Magnitka. In quantity they greatly exceed the reserves of these known deposits. The Kustanai ore lies near to the surface and can be won by the open-cast method.

The Twentieth Session of the Party laid down provisions for the construction and commissioning of a Sokolov-Sarbai ore treatment plant with a capacity of 10 million tons of crude iron ore. By 1960 5.6 million tons of treated ore must be produced.

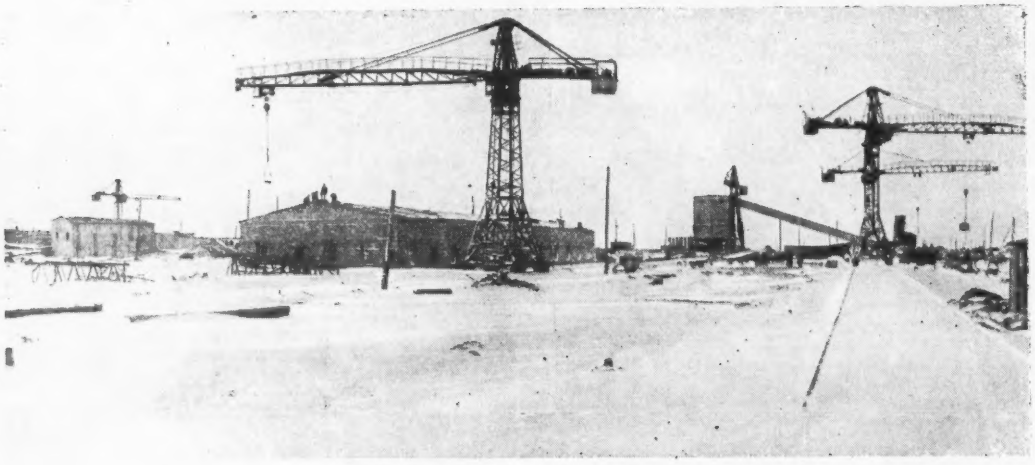
The workers of the Kustanai region, with its limitless expanses of virgin deposits, are faced with a fresh a task in the sixth Five-Year Plan, namely to lift from the bosom of the earth the rich iron-ore deposits.

The Sokolov-Sarbai works is an integrated undertaking for the winning and enrichment of iron ore, together with electric power stations for its own requirements, its own water economy, and the manufacture on site of a large measure of ancillary components for mine transport equipment.

The Sokolov and Sarbai sites carry mainly primary magnetite ores, the reserves of which comprise 93% of all the reserves on the site. The remainder comprises martite and sub-martite ores.

The thickness of the iron-ore deposits is 280 m, extending for more than $3\frac{1}{2}$ kilometers.

The ore deposits lie between rock, having a top covering of marine and river deposits – sand, clay and sandstone. The geological conditions of this layer are extremely complex, having thick water bearing levels with a high head. The sand has quicksand characteristics, setting ususually difficult conditions for carrying out the work of removing the over-burden.



Works construction site.

The works take in the Sokolov and Sarbai quarries, which are 9 kilometers apart, together with a number of ore treatment plants, a 50 MW electric power station, an engineering repair plant, the Sergeev and Karatamar reservoir, the Kustanai-Tobol railway and other industrial installations. When the plant is working at full capacity its internal transport load will comprise 240,000 tons per day, or 4,800 50-ton railway wagons. 120 heavy six-axle electric locomotives will be working simultaneously, together with approximately 1,000 60-ton automatic dump trucks.

The over-all length of the railway on the plant site will be 300 km. Every day up to 50,000 tons of various grades of iron-ore will be despatched to Magnitogorsk, Chelyabinsk and other Ural plants.

More than seventy heavy excavators will be operating on the quarries, including walking drag-lines with a scoop capacity of 14 m^3 and a reach of 75 m. For the stripping work on the Sokolov quarry a multi-caterpillar excavator DS-1000 is already set up and this machine can remove 1,000 cubic meters of earth per hour. The size of the machine is such that a railway train can freely pass under it. In order to commence the work of winning the ore it is necessary to run outer access ways and inner intersecting trenches in order to open up access

to the ore and to put under way stripping of the over-burden. A particular difficulty is met with in stripping the third layer of barren rock, consisting of fine saturated sand 30 m thick. The walking drag-line will operate here with a scoop capacity of 14 m^3, standing over the sand as on a base of a 10 meter layer of opoka and clay.

The ore will be quarried by blasting with drills over the whole height of the face (15-20 m). The driven ore will be loaded into railway wagons or dump trucks and despatched to the treatment plants.

The richest ores, uncontaminated by barren rock and sulfur are despatched direct from the excavators to the crushing grading plant. The lumpy ore is then passed to the stock-pile and onto the consumers.

The blast-furnace ores will contain 58-59% of iron and the open-hearth furnace ores over 60%.

The main mass of Sokolov and Sarbai ore—approximately 90%—has inclusions of barren rock and contains sulfur. It is passed to the blast furnace only after fine crushing and magnetic separation (sintering will be carried out at Magnitogorsk and Chelyabinsk).

The magnetic separators comprise machines with specially magnetized rotating cylinders. The ore particles are shaken on to the cylinders and conveyed around the surface of the cylinder to openings in the equipment over which they are released and drop off onto the conveyer belt. The barren rock particles, not being attached to the cylinder by magnetic traction, fall into a chute.

With the removal of the barren rock particles the ore is enriched and the iron content rises from 46-48 to 58-60%.

Loading rock onto railway flat cars.

A number of problems have to be resolved at the works, connected first with the water supply for the production process relating to the technical and power requirements, together with the personnel requirements in drinking water, and second the drainage of groundwater impeding quarrying and at the same time representing a danger to the workers and injuriously affecting the equipment.

In 1957 the Sergeev reservoir will be put into commission on the river Tobol, and subsequently the Karatomarsk reservoir having a capacity of 500 million m^3.

In order to solve the second problem work will be carried out for lowering the water table. Around the site of the quarry, bore-holes will be drilled with special machines carrying at various depths submerged pumps

for pumping out the underground water. The groundwater is acrid and is unsuitable for industrial or drinking purposes. The work for lowering the water table is difficult and expensive.

The Council of Ministers of the USSR has fixed dates for commissioning the units. In 1957 50% of the Sokolov quarry has to be commissioned, together with the steam power station, the dry magnetic separation plant and—on full capacity—the sinter, blast furnace and open-hearth ore plant.

The works are to deliver the first ore in 1957.

Construction of the works will cost $2^1/_2$ billion rubles. Approximately 1,500 million cubic meters of excavation work has be be carried out—as much as the earthwork on the Kuibyshev hydraulic construction. Approximately 1 million m^3 of concrete and reinforced-concrete will be placed and more than 100 thousand tons of metal construction erected. The works will be provided with first-class Soviet equipment embodying automatic control of the basic processes at the ore treatment plants.

Assembling multi-ladle excavator.

In drawing up the equipment for the works the administration has expended more than 200 million roubles in the construction of living accommodation, schools, hospitals, clubs and welfare. In every dwelling of the new town of Rudny being constructed on the banks of the Tobol there will be central heating, hot and cold water. Gardens, open spaces and a swimming pool will render the town good-looking and comfortable.

Erecting the reinforced-concrete structures of the factory.

CONTINUOUS TEMPERATURE CONTROL IN THE COMBUSTION ZONE OF A BLAST FURNACE

I. F. Koval and A. Ya. Kravchenko

The volume of the combustion zone has a considerable influence on the uniformity of descent of the charge. Blast furnacemen, therefore, always strive to set up a sufficiently broad combustion zone without reducing the temperature in the hearth. The parameters of the combustion zone (with constant coke quality) are regulated by the variation in blast temperature, quantity of blast or quantity of moisture in the blast.

At the present time the thermal condition of the furnace and variation in driving of the thermal processes are determined principally at the tuyeres, from visual observation, breakdown and composition of the slag together with the temperature and composition of the pig-iron.

Control and regulation of the thermal condition of the furnace make for better driving and higher productivity. The means for observing the thermal condition of the hearth and the methods for expressing quantitatively the heat of the furnace are, however, by no means perfect. As a result of errors subject to the individual peculiarities of the operator, incorrect solutions are adopted, which give rise to irregular performance of the furnace. All this points to the necessity for continuous control of the thermal condition of the hearth and, in the near future, the installation of automatic control for maintaining the optimum degree of driving of the hearth.

For the purpose of establishing this type of control continuous observations of the temperature in the combustion zone of the blast furnace have been organized. The measurements were carried out on the Kostogryzov thermometer system through the tuyeres of the blast-furnaces (Fig. 1).

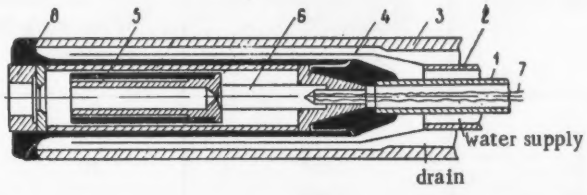


Fig. 1. Thermometer for measuring the temperature in the combustion zone:
1) Internal copper pipe; 2) copper dividing tube; 3) outer steel tube; 4) brass dividing cap; 5) collector tube; 6) steel core with thermopile; 7) leads from thermopile for connecting to the potentiometer; 8) stainless steel clamping nut.

In order to assess the temperature in the combustion zone two thermometers were set up in Tuyeres No. 2 and 8 (located diametrically opposite). Both thermometers provided mean indications on the potentiometer.

The thermometer operated on the basis of heat radiated by the gases (mainly CO_2) in the hearth of the furnace. The heat was collected through a tube and the temperature transmitted from a thermocouple (copper-constantan) to the potentiometer. The thermocouple was set up in the nozzle of the tuyere (Fig. 2) in such a way that the end fell short of the mouth of the tuyere by 50-60 mm. The tuyere protected the instrument from

damage by the blast furnace materials. The thermometer was cooled by water from the town mains. The instrument was installed strictly in line with the axis of the tuyere, since there are different temperatures at different points in the combustion zone.

Observations indicated that the temperature in the combustion zone even with smooth air supply and constant blast temperature does not remain constant and varies mainly as a function of the gas permeability of the burden. It was noted that even discharging a fresh batch of material onto the furnace top gave rise to an increased temperature in the combustion zone and, vice versa, the temperature fell as the stockline descended. A greater density of the stock column gave rise to a higher temperature in the combustion zone.

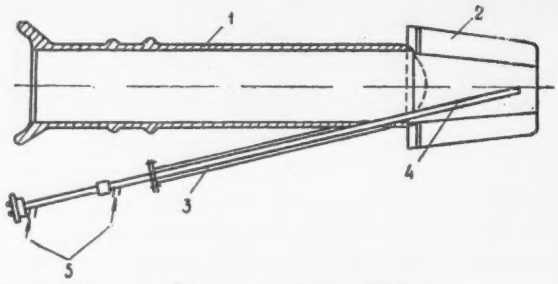


Fig. 2. Diagram of thermocouple installation:
1) Nozzles; 2) tuyere; 3) support and flange; 4) thermocouple; 5) cooling water inlet and drain.

The thermometers showed that in smelting Bessemer iron with a content in the burden of 70% crude ore and 30% very fine agglomerate the optimum temperature in the combustion zone was approximately 1850°. On this limit being exceeded there was delay in the delay in the descent of the burden (accompanied sometimes by hanging), and on falling below this maximum there was rapid descent, with further cooling of the furnace.

The table provides figures relating the temperature at the tuyeres with the time taken for descent of the stockline. The figures show that the higher the temperature at the focal spot the longer is the time for descent of the charge.

charge number	temperature at the nozzles °C	time for descent of the stock-line, minutes*
1	1850	9
2	1850	10
3	1850	
4	1830	8
5	1870	14
27	1900	18
28	1860	16
93	1830	8
94	1820	8
95	1800	6

* Time counted for descent of one charge to the steady stockline.

Figure 3 illustrates the principal operating charts for one furnace on July 26, 1955. In the course of that day fluctuations in the pressure and temperature of the blast were observed. The temperature in the combustion zone was high, from 10:30 p.m. to 10:45 p.m. for example, reaching 1980°, which coincided with a reduced descent of the burden. By 4 a.m. descent of the burden improved but in the periods of high temperature in the combustion zone the rate of descent of the charge fell off considerably. From 4 a.m. to 6. a.m. the temperature in the combustion zone rose to 1900°, descent of the burden became very irregular and hold-ups and break-throughs were observed. A typical feature was that the temperature in the combustion zone began to increse from 4 a.m. while the hot blast pressure even fell slightly (by 0.04 atm.), and the air supply began to fluctuate only at 4:45 a.m. The furnace crew, not knowing the thermal condition of the hearth, began to raise the temperature of the blast, taking it up to 800° by 4:45 a.m., thus, causing hanging of the burden and interruptions in performance.

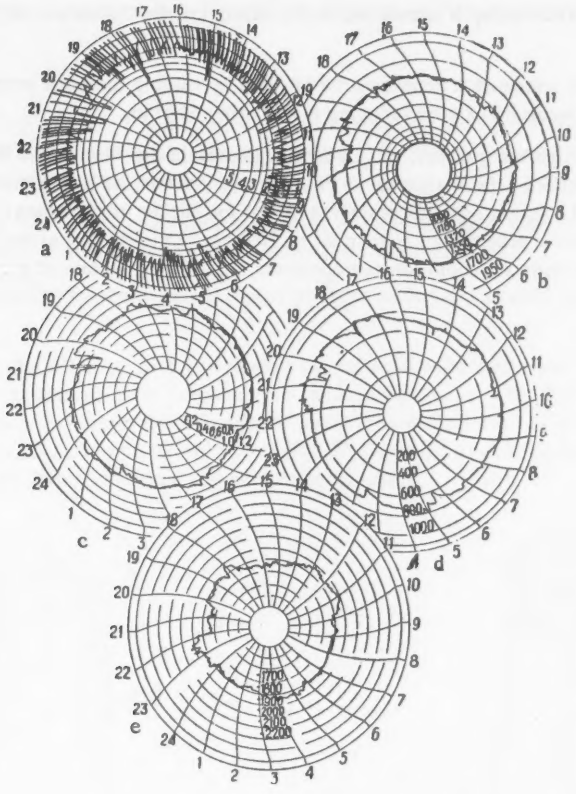


Fig. 3. Charts of furnace operation, July 26, 1955.
a) Stock-line level; b) blast consumption; c) hot blast pressure; d) hot-blast temperature; e) temperature in the combustion zone.

In the period from 10:30 a.m. to 1 p. m. the temperature in the combustion zone rose to 1960°. During the same time there was a delay in the descent of the burden. During the period from 5:20 p.m. to 6:15 p.m. the temperature rise in the combustion zone resulted in the air supply being reduced, the blast pressure being increased and the rate of the burden being reduced. A reduction in air supply at 8:45 p.m. from 1550 to 1370 m^3/min resulted in a reduced temperature in the combustion zone to 1800°. This tardy measure, however, did not obviate forced sagging(at 9 p.m. and 9:30 p.m.).

Delayed descent of the burden and hanging were associated with an increase in the temperature of the combustion zone to 1990°. The furnace was brought back to smooth operation only be lowering the temperature.

The following conclusions can be drawn.

A thermometer for continuous temperature mea surement in the combustion zone makes it possible to follow continuously the heat of the hearth and the variations in the thermal performance. On the basis of the instrument readings it is possible to adopt measures providing a constant temperature in the hearth.

The temperature in the combustion zone is closely related to the gas permeability of the burden. Increasing the gas permeability produces a reduction in the temperature of the combustion zone, while reducing the gas permeability leads to a higher temperature in the combustion zone.

As the temperature rises above the established value it is necessary immediately to take measures for reducing the temperature,otherwise delay is introduced in the descent of the materials, or hanging of the burden occurs.

The temperature in the combustion zone must be lowered by varying the blast temperature or adding steam to the blast or, again, in the extreme case by reducing the quantity of blast.

The employment of a high top pressure, self fluxing agglomerate and humidified blast have had a certain effect in blast furnace production. The maintenance of a constant heat at the tuyere provides the possibility of obviating tight operation and hanging and will provide a means for further improvement in the furnace operation. The installation of thermometers provides an assessment of the heat at the tuyeres as read off on the instrument and not "by sight", thus providing a smooth furnace operation, and the production of pig-iron with a more uniform chemical composition.

MEASURES FOR SPEEDING-UP BLAST FURNACE OVERHAULS

V. N. Mashyanov

Chief Engineer, Chelyabinsk Construction Department "Ulraldomnaremont"

At the Chelyabinsk steel works a quick overhaul was carried out in August, 1956 on second-class blast furnace No. 2 with a useful volume 930 m^3. The furnace has an inclined skip charger, a double-bell charging gear and a charge distributor with water seal.

The following work was carried out during the period of the overhaul: replacement of the refractory lining of the tuyere zone, the bosh, cylinder and stack; encrustations were removed.

In the preparatory period for the overhaul a plan of organization was worked out together with a chart for carrying out the work.

In accordance with the plan the gear was assembled, stocks of refractories and metal structures were laid up, metal constructions and castings were prepared, the brick was graded out and the cooling boxes were pressed. On August 17th, the furnace was blown down to a level 1 m above the tuyeres. Slightly below the throat-protection plates there was encrustation to the volume of 300 m^3 (Fig. 1). After lowering the charge, breaking-up of the encrustation was carried out to the third row of the shaft coolers.

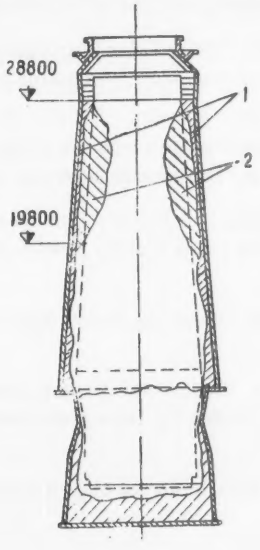


Fig. 1. Encrustation in the furnace:
1) Lining; 2) encrustation.

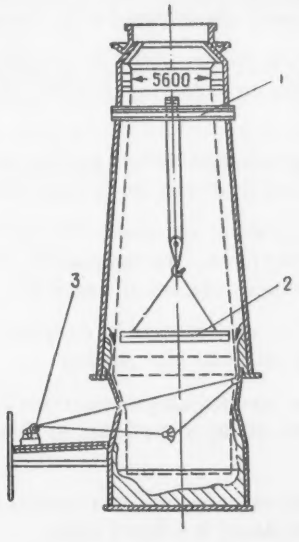


Fig. 2. Scraper installation for removing debris:
1) Deck; 2) suspended platform; 3) scraper installation.

For removing the encrustation and the remaining charge from the furnace in the tuyere zone a gap was cut 1.8× 1.7 m on the side of the sloping bridge. A second opening 1.8 × 3.0 m was cut in the bosh. Large pieces of crustation were removed through the gap by means of winches.

The shaft lining was broken up with pneumatic hammers from a suspended platform which was lowered as required.

A deck was set up under the throat-protection sheets. The hoisting tackle for the suspended platform was fixed to the deck.

Debris from the furnace was removed by scraper to the wagons (Fig. 2).

The coolers of the bosh and tuyere zone were dismantled by a motor hoist and removed from the furnace by means of winches through the gap in the tuyere zone.

Dismantling took six days.

The coolers of the bosh and tuyere zone were erected with the help of a crane (Fig. 3) equipped with gear providing rotation through 360°. The crane facilitated the work and increased the labor productivity of the erectors. Twenty cooler units were set up in eight hours in place of 6-8 units without the crane.

The coolers were brought up to the furnace along a trestle on a bogey (Fig. 4), erecting them with pipes downwards; this cut out the necessity for tilting the coolers.

Erection of the bosh and tuyere zone coolers was completed in two days four hours; thirty-five man-shifts were employed in the erection.

Parallel with this, overhaul was carried out on the case, tuyere zone, bosh and shaft, together with other erection work.

The jacket of the gas down-comers was overhauled from a suspended platform which was moved along as required by a motor winch.

Before commencing overhaul the brick was assembled near the work and graded. The brick was delivered from the stock by means of 500 mm belt conveyors with automatic transfer to inclined conveyors by means of which the brick was transferred to bucket conveyors and then to the work site by means of 200 mm wide conveyors. Loose material was delivered to the furnace in buckets with a capacity of 0.75 m^3 on a chain conveyor.

The mortar was prepared under the platform of the casting bay and delivered along two mortar lines by means of compressed air employing air tanks and mortar pumps. The shaft was lined from a suspended platform (Fig. 4).

A multi-stage platform made it possible to coordinate assembly of the coolers with the lining of the lower stages of the shaft and to obviate the dead periods inevitable for raising the single-stage suspended platform.

In lining the shaft use was made of an air-hardening mortar consisting of a fireclay alumina mortar from the Pyshma refractory plant. The composition of the mortar was as follows: 80% fireclay powder, 5% clay, 15% alumina together with an addition of waterglass.

The mortar was characterized by complete homogeneity, ran freely through the mortar pipes and bonded well with the brick two hours after placing.

The use of an air-hardening mortar made it possible to increase the thickness of the ring joints to 2.5 mm, without lowering the quality of the lining, at the same time considerably raising the labor productivity of the refractory workers.

The mean output for one refractory worker amounted to 1.6 m^3 of shaft lining for eight hours work. 420 m^3 of shaft lining were placed in 4 days 8 hours.

Lining of the bosh and tuyere zone was carried out on a fireclay mortar. The maximum output for one refractory worker in lining the tuyere zone amounted to 2.2 m^3 per shift. The work of lining the bosh and tuyere zone was completed in 1 day 22 hours.

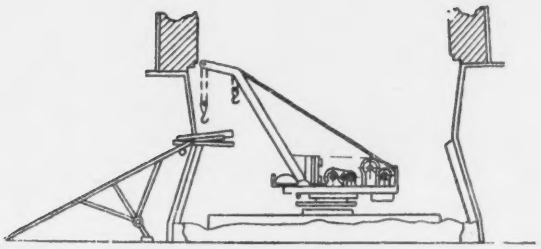


Fig. 3. Crane for erecting the coolers.

The furnace was blown in on August 29th. The total time taken for the overhaul was 12 days. During the overhaul 775 men were occupied on the fifth day and 651 men on the tenth day.

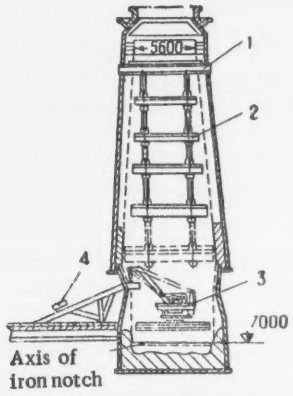


Fig. 4. Diagram of erection structures.
1) Sub-segment deck; 2) five-stage suspended platform for lining the shaft; 3) full-circle jib crane for erecting coolers; 4) bogey for delivering coolers to furnace.

The following measures are effective for the purpose of reducing the overhaul time on blast furnaces:

1) the delivery of coolers to the furnace by bogey along a trestle;

2) the erection of the hearth, bosh and tuyere-zone coolers with the aid of jib cranes;

3) the employment of a hanging platform manipulated by motor hoists for the overhaul of the gas down-comers;

4) the use of scrapers for clearing the hearth of debris;

5) the use of an air-hardening mortar for the lining;

6) the erection of multi-stage platforms in the furnace shaft.

In addition it is necessary to re-design the throat erection beams, increasing their lifting capacity to limits which will allow for the assembly of charging gear in large units: 1) large bell and cone, and 2) gas valve, rods and small bell.

IRON AND STEEL IN 1956

(From the Data of the Central Statistical Bureau of the Council of Ministers USSR).

The undertakings of the Ministry of Iron and Steel USSR fulfilled the annual plan for 1956 to 99.6%. The output of pig-iron was 35.8 million tons (107% of the 1955 figure), steel 48.6 million tons (107% of the 1955 figure), rolled steel 37.8 million tons (107% of 1955), steel tubes 3.8 million tons (108% of 1955) and iron-ore 78 million tons (109% of 1955).

An important condition of increasing industrial production in 1956 was the further improvement of manufacturing technology and the utilization of existing means of production. In the plants of the Ministry of Iron and Steel USSR the utilization of the useful volume of blast furnaces improved in 1956 by 3% over 1955, while the steel output per square meter of furnace hearth also rose by 3%. At the same time the existing reserves at the plants of the Ministry for increasing the production of ferrous metals were inadequately employed. Throughout the Ministry as a whole the assignments for 1956 for the utilization of blast furnaces and open-hearth furnaces were not fulfilled. There was a considerable short-fall also in the plan for lowering the cost of production.

Further work was carried out in iron and steel manufacture in the introduction of new techniques, the use of oxygen for steel smelting in open-hearth and electric steel smelting furnaces, in converter production and in the smelting of blast furnace ferrous alloys, the use of heat resisting refractories, the establishment of the process of continuous steel casting and the employment of vacuum in metal casting, together with automatic control of the production processes. At the same time the Ministry of Iron and Steel USSR did not achieve fulfillment of the assignments for the introduction of oxygen plants, automatic control of rolling mills and mechanized grading of iron and manganese ore and rock in quarry driving.

In 1956 large blast-furnaces were brought into operation at the Dneprodzerzhinsky and the Stalin works, in addition to two open-hearth furnaces with a capacity of 400 thousand tons of metal per year at the Voroshilovgrad steel plant. At the same plant there was built a new blast furnace which was tapped for the first time in the early days of January, 1957. A plant was brought into operation for the production of cold-rolled sheet at the Magnitogorsk works. However, during the past year the plan for bringing into operation further productive capacities was underfulfilled. Six rolling mills, one blast furnace and two open-hearth furnaces, together with two Bessemer converters, all specified in the plan, were not commissioned.

STEEL SMELTING

THE SMELTING OF STRUCTURAL CHROME STEEL IN 380-TON OPEN-HEARTH FURNACES

B. G. Petukhov

Chief Research Engineer, Central Laboratory Kuznetsk Works

In recent years, as a result of the establishment of the 380-ton furnaces and the advancement of the metal smelting technology, nearly all high-grade steel (medium and high carbon, tube, cable, instrument, manganese etc.) is smelted in this type of furnace.

Up to the middle of 1956 the Kuznetsk steel workers smelted chrome steel (20X, 40X) in 190-ton open-hearth furnaces and poured it through an intermediate ladle into moulds broadening out at the top, with iron-lined shrink holes. At the present time, part of the grade 40X and 20X metal is smelted in 380-ton furnaces and is poured from the top. At the same time the main attention is turned towards achieving normal slag and temperature regimes, and to the rate of carbon burn-up and the quality of the smelted steel.

In smelting in large furnaces the charge consists of 62-68% molten iron, 32-38% steel scrap, 10.6-12.3% iron-ore and 4.2-5.9% limestone. It has been established that on increasing the limestone consumption in the charge from 43 to 59 kg/t the smelting time is increased on average by 45-60 minutes.

The charging time of loose materials with heating-through of each layer amounts to 45-60 minutes.

The metal scrap is charged as quickly as possible, attaining normal heating before charging the liquid iron. The heat liberation per unit volume of combustion space during the over-all charge amounts to 28-30 million cal/hr.

The charging system adopted provides good slag formation, active boiling and mixing of the bath, and speeds up the process of smelting of the charge. All this supports active dephosphorization and desulfurization of the metal.

The iron ore during the period of ore boiling is introduced into one receptacle, (not more than two moulds). The quantity of iron ore introduced during this period is determined by the carbon content in the metal according to the smelting, oxidizing capacity of the furnace and the temperature of the metal. The quantity of slag skimmed in the dephosphorization period (0.5-1 ladles) depends on the phosphorus and sulfur content during smelting. After drawing-off the slag the P content in the metal is usually not more than 0.018%. For laying-on fresh slag, lime is added (3-5 moulds, depending on the phosphorus content in the metal after running off the slag) together with bauxite.

In the finishing boil period iron ore is added as required to a quantity not greater than 0.75 moulds and lime not more than 1 mould after 45 minutes following boiling. Ore is added for accelerating smelting of the lime and converting the bath to a finishing boil.

The smelts are carried out without adding ferromanganese with the residual manganese in the metal 0.04-0.08%.

The carbon content during smelting and the lime additions have a considerable influence on the finishing period. In the smelting of grade 40X steel in 380-ton furnaces with an increase in the carbon content during

smelting from 1.10 to 1.40% the smelting period is increased on average by 60 minutes while on smelting grade 20X steel and raising the carbon content from 0.80 to 1.0% the period is increased by 40-50 minutes. Unjustified excessive lime additions in the finishing period also increase the final melting period by 20-30 minutes.

It is desirable that the carbon content in smelting in the case of grade 40X should be 1.0-1.15% and grade 20X steel 0.80-0.85%.

The heat liberation in the finishing period should be 22-25 million cal/hr.

The smelting time in the smelting of chrome steel in 380-ton furnaces is increased basically on account of the prolonged charging and smelting and on account of the high carbon content during smelting (table).

Characteristics of the Chrome Steel Smelt

Characteristic	40 X steel		20 X steel	
	Furnace capacity, tons			
	380	190	380	190
No. of smelts investigation	12	12	10	12
Content % in the metal: during smelting C	1.17	1.28	0.86	1.12
during smelting P	0.039	0.017	0.034	0.019
on completing dephosphorization C	0.69	0.86	0.51	0.66
on completing dephosphorization P	0.017	0.009	0.017	0.011
Carbon burnup, %C/hr.				
dephosphorization	0.34	0.62	0.40	0.59
laying-on slag	0.27	0.23	0.21	0.25
finishing boiling	0.22	0.29	0.18	0.26
Duration of period, hr-min:				
dephosphorization	1—21	0—47	0—48	0—45
laying-on slag	0—39	0—50	0—44	0—46
finishing boiling	1—13	1—21	1—17	1—25
reduction	0—45	0—38	0—45	0—33
overall smelt	12—42	9—20	13—00	9—50
Overall loss of reducing elements, %of total introduced:				
Mn	26.6	23.4	21.3	26.6
Si	34.1	42.0	37.2	44.5
Cr	17.0	19.8	18.8	21.2
Mean metal temp. before reduction,* °C	1608		1614	

* Measured on an immersion platinum – platinum rhodium thermocouple of Kuznetsk design.

The mean rate of carbon burn-up during the dephosphorization and finishing boiling period in smelting chrome steel in 380-ton furnaces is less than in smelting in the 190-ton furnaces. This is caused by the exceptional difficulty of heating the metal through in large furnaces. During the period of laying-on the slag the speed of decarbonization in the furnaces is practically the same for both types. Despite the lower carbon burn-up rates during the finishing boil (0.18-0.22% C per hour), active boiling of the metal is achieved in the large furnaces.

The consumption of iron ore has a considerable influence in raising the rate of carbon burn-up, with a good thermal regime of course. In smelting grade 40X steel on raising the iron ore consumption from 90 to 135 kg for 0.01% C, the mean carbon burn-up rate rises from 0.25 to 0.31% C per hour. This is also typical of grade 20X steel.

The heating rate of the metal during the finishing boil period was: for grade 40X steel-0.76-1.75° per minute (on average 1.26° per minute) and for grade 20X steel-0.79-1.55° per minute (on average 1.18° per minute).

The necessary metal heating rates are achieved by reason of good mixing and consequently good thermal conditions in the bath with high carbon burn-up rates.

The nature of the change of the elements in the metal and of the basic characteristics of the slag in the finishing process are practically identical for the different furnaces (see diagram). Outstanding features in smelting grades 40X and 20X steel in 380-ton furnaces are lower carbon content in the metal, the higher basicity of the slag during the finishing boil, the higher iron oxide content in the final slag, reduced manganese content and higher phosphorus content in the metal together with the lower rate of carbon burn-up in the period of dephosphorization and the finishing boil.

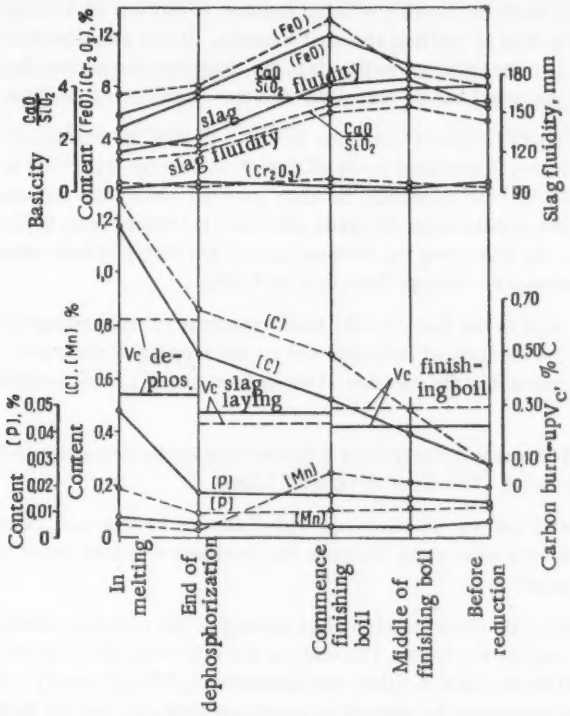


Fig. 1. Variation in the mean concentrations of C, Mn and P in the metal and in the slag components during the smelting of grade 40X steel; solid lines: in 380-ton furnaces; broken-lines: in 190-ton furnaces.

Analysis of the variation of the metal and slag composition during finishing point to the fact that in smelting grades 40X and 20X in large furnaces there is a much higher phosphorus content in the metal than when melting in 190-ton furnaces. This is explained by the higher phosphorus content in these melts (for grade 40X steel 2.6 times and for 20X steel 1.9 times greater than in small furnaces). As a result of careful dephosphorization the phosphorus content in the metal drops to 0.017% at the end of dephosphorization, which provides a phosphorus figure before reduction of 0.010-0.013% and in the finished steel 0.023-0.027%.

The basicity of the slag is kept high during blowing (2.4-3.2) but this does not destroy the intensity of boiling in the bath. In the finishing boil the slag is freely running; the slag yield before reduction is 120-180 mm.

In the case of high basicity slags the phosforus content in the metal during the process of the finishing boil does not rise up to the reduction period but remains constant or falls off slightly and then as in the case of melting in the 190-ton furnaces some increase in the phosphorus content is observed.

The smelting technology adopted for chrome steel in 380-ton furnaces does not influence the degree of oxidation of the metal over a wide range of carbon concentration. The oxygen content in the metal before reduction (with 0.25-0.30%C) variesover the range 0.013-0.025% and on lowering the carbon concentration to 0.08-0.10% it rises to 0.040-0.050%.

Preliminary reduction when melting in 380-ton furnaces was carried out in the following ways: a) by blast furnace ferrosilicon and ferromanganese; b) by blast-furnace ferrosilicon, ferromanganese and silicomanganese; c) by silicomanganese and ferromanganese and d) by silicomanganese alone. These varied measures for preliminary reduction were not the result of process codes of instruction but the availability at the works of one or another reducing agent at the given time.

As shown by the table, the consumption of Si and Cr in melting in the 380-ton furnaces was 2-8% less than on melting in the 190-ton furnaces. The consumption of chromium was different for different grades of chrome steel. In the case of grade 20X steel, in melting in large furnaces it was less on average by 2.3% and in melting grade 40X steel higher by 3.2% than in melting in small furnaces. It was established that the consumption of Cr and Mn rises with increase in: FeO content in the slag, the time between adding the ferroalloy and tapping the smelt, the temperature of the metal before reduction and the basicity of the final slag.

For example, with increased FeO before reduction from 7.5 to 10% the chromium consumption (in % by weight of the liquid steel) increases in smelting grade 40X steel on average from 0.15 to 0.22% and in smelting grade 20X steel from 0.10 to 0.17%. On increasing the time between adding the ferrochrome and tapping from 30 to 40 minutes the expenditure of chromium for grade 40X steel is reduced from 0.92 to 0.80% and for grade 20X steel from 0.85 to 0.75%. On increasing the temperature of the metal before reduction from 1605 to 1630° the chromium expenditure increases on average from 0.10 to 0.20%.

When reducing chrome steel in the furnace with ferromanganese or silicomanganese the manganese consumption is related closely to the duration of reduction and on the tapping of the smelt. It is therefore necessary to turn particular attention to restricting the duration of the reduction and to the condition of the steel tapping opening.

The metal was poured downwards directly from a double-stopper steel-pouring ladle into a mould broadened at the top with ceramic shrink-holes. The ingot weighed 7.2 tons.

For pouring grade 40X steel use was made of cups with diameter 35 mm and, for grade 20X steel, diameter 33-35 mm. Cups of these diameters were quite adequate for providing a normal jet of metal without forming a scale on the surface of the ingot.

The shrink-holes were filled up immediately before passing to the adjacent mould, so that adhering scale would not be carried into the body of the ingot. The body of the ingot was filled in 2-4 minutes and the shrink-hole 40-60 seconds. The head of the ingot is filled with lunkerite (1.5-2 kg/t steel). In order to avoid scab as the jet strikes the bottom of the mould the stopper is opened gradually to half jet during the course of 3-4 seconds and is then fully opened.

The ingots are kept in the pouring bay for 80-100 minutes. The chemical composition of the melted metal varies over the following ranges in %:

	C	Mn
grade 40X steel	0.35–0.42	0.50–0.70
grade 20X steel	0.16–0.18	0.50–0.64
	Si	Cr
grade 40X steel	0.21–0.33	0.90–1.03
grade 20X steel	0.22–0.31	0.76–0.91

Mechanical tests indicate that chrome steel melted in 380-ton furnaces has better mechanical characteristics than those provided for in the GOST Russian standard.

For example, the creep limit of grade 40X steel is higher on average by 19 kg/mm^2, the ultimate strength higher on average by 7 kg/mm^2, the relative elongation is higher on average by 5%, lateral contraction is greater on average by 10% and toughness greater on average by 1.5 kgm/cm^2. The same can be stated with regard to grade 20X steel.

Comparison of the quality of chrome steel shows that the metal melted in large furnaces is in no way inferior to metal melted in the 190-ton furnaces.

Results of smelts carried out at the Kuznetsk steel works provide the following conclusions:

1. The smelting of grades 40X and 20X structural steel in 380-ton open-hearth furnaces is possible and does not present any special difficulties.

2. The mechanical properties of metal melted in the 380-ton furnaces do not differ from the metal melted in 190-ton furnaces and surpass the requirements of the technical specifications. The macrostructure of the metal is good.

3. Calculations indicate that, for example, a monthly melt of 30 ladles of chrome metal (40X and 20X) in 190-ton furnaces requires an average 290 hours and for melting in 380-ton furnaces 195 hours; in other words, the use of the open-hearth furnaces is reduced by 95 hours. In the time thus saved further melting of approximately 1800-1900 tons of metal is possible.

PROTECTION OF RADIATION PYROMETERS FROM DAMAGE BY THE FURNACE FLAME

V. M. Studentsov

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At the present time use is being made successfully on the open-hearth furnaces of continuous arch temperature measurement in which radiation pyrometers are directed onto the furnace arch through a metal sighting tube with diameter 150 mm built into the rear wall of the furnace at an angle to the arch.

Two radiation pyrometers are as a rule set up on each furnace, symmetrically located along the length of the furnace. The furnace flame, striking partially onto the sighting tube, is blown back by compressed air introduced from inside the tube. Fluctuations in the compressed air pressure or sudden interruption of the air from the factory main is a disturbing factor in the normal operation of the instruments. A sharp rise in the air pressure leads to cooling of the sighting section of the arch and consequently to an increase in the measurement error; lowering the pressure or cutting off the supply of compressed air leads to the flame striking the sight tube and frequently to burning away of the radiation pyrometer.

The pneumatic gate employed by us automatically avoids the flame striking the sight tube when the pressure in the compressed air main falls below 2.5 kg/cm^2 and provides normal blowing of the flame with compressed air pressures over the range 6-2.5 kg/cm^2. The principle of operation of the automatic gate (Fig. 1) is as follows.

The compressed air entering the cylinder compresses the piston to the left and passes through an opening, diameter 6 mm, in the sight tube. The flame falling onto the tube from the furnace is blown off by the air. When the compressed air main falls to a pressure below 2.5 kg/cm^2 the spring presses the piston to the right to the initial position. At the same time the check piece is drawn into the opening of the face piece and liberates the slide, which under the action of its own weight rotates on a hinge, thus closing the end of the sight tube. The sight tube is closed by the slide before the piston closes the 6 mm diameter hole. In this way the flame is prevented from striking the radiation pyrometer.

The hole in the cylinder wall passes sufficient air for blowing off the flame and the furnace arch is not cooled at the measurement point even with maximum air pressure (6 kg/cm^2).

The sight tube consists of two sections: the outer with diameter 125 mm and the inner with diameter 150 mm going up to the furnace wall (Fig. 2). The outer tube section is welded to the shell of the furnace and the 150 mm tube section is fitted on to it, this being usually done during the cold overhaul of the furnace.

The water-cooled tuyere is fitted to the outer face of the tube with an articulated joint, and the radiation pyrometer is fitted onto this by means of a special device (Fig. 3). By twisting the handle of the mechanism the radiation pyrometer can be fixed in place along the axis of the sight tube and on rotating the grip in the reverse direction it can be quickly and readily removed.

For measuring the checker-top temperature of open-hearth furnaces the radiation pyrometer is set up in an uncooled tuyere of special construction (Fig. 4). There is a slide in the tuyere cutting off the hot gas from the furnace to the pyrometer in case of a drop in the pressure of the compressed air used for blowing the hot gas from the instrument. The compressed air from the main enters a nozzle having an exhaust channel of 5 mm diameter.

The air jet leaving the nozzle strikes the slide, which raises a counter-weight on rotating around its axle. The gate chamber is enclosed in a housing fixed by means of two pins and nuts to the sight tube. The radiation pyrometer is fitted as shown in Fig. 3. When a pressure drop occurs in the compressed-air main the counter-weight falls and the slide covers the channel of the sight tube. The sensitivity of the slide is adjusted by moving the counter-weight along the arm.

The temperature of the arch of soaking pits for heating ingots can be better measured by means of the radiation pyrometer sighted onto the bottom of a refractory cup, set up in the furnace arch. The bottom of the cup projects into the furnace and is heated to the temperature of the arch. The radiation pyrometer is situated in the tuyere with water cooling and is fixed by means of the device shown in Fig. 3. Fixing the tuyere separately from the refractory cup does not give advantageous results since the arch is distorted through the sharp temperature variations and there are frequent deviations in the focus of the radiation pyrometer onto the bottom of the cup.

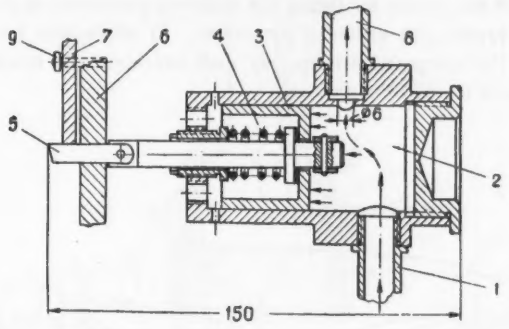

Fig. 1. Diagram of the automatic air gate.
1) Fixing bracket; 2) cylinder; 3) piston; 4) spring; 5) check-piece; 6) face; 7) slide; 8) pipe; 9) hinge.

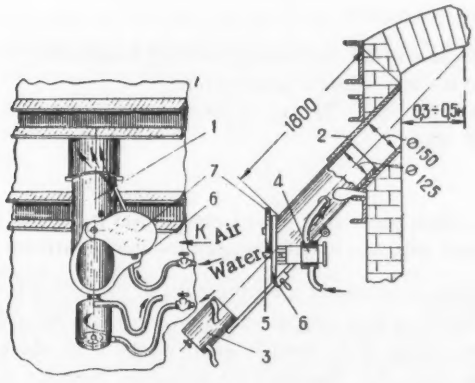

Fig. 2. General view of the installation for temperature measurement of the arch of an open-hearth furnace with radiation pyrometer.
1) Sight tube (outer) diameter 25 mm; 2) sight tubes, built into the furnace wall; 3) water-cooled tuyere for radiation pyrometer; 4) automatic air gate; 5) check-piece; 6) face; 7) slide.

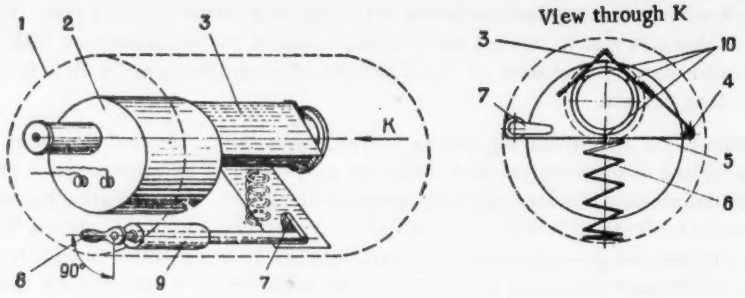

Fig. 3. Diagram of the device for fitting the radiation pyrometer in the tuyere.
1) Water-cooled tuyere; 2) radiation pyrometer; 3) angle iron (45 X 45 X 4 mm); 4) hinge; 5) compression strap; 9) shaft bearings; 10) bearing planes, fixing the position of the pyrometer

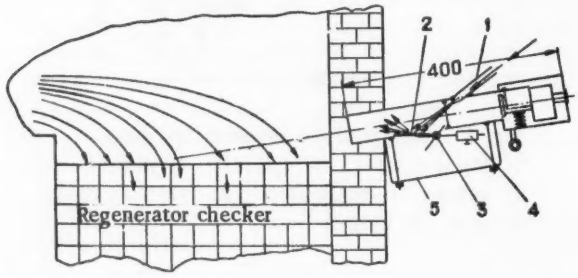

Fig. 4. Diagram of the radiation pyrometer installation for measurement of the top checker temperature.
1) Nozzle; 2) slag; 3) axis of rotation of the slide; 4) counterweight; 5) case.

Good results are obtained when the fixing is so arranged that the end of the sight tube of the tuyere enters the channel of the refractory cup, and is self-centered towards it without external adjustment (Fig. 5).

In order to protect the radiation pyrometer the sight tube of the tuyere is closed by means of a slide which in the normal open position is held by a thin copper wire of 0.3-0.5 mm diameter. The wire consists of two pieces soldered with lead. If the bottom of the bowl is burnt-out the lead solder is melted as a result of the higher temperature and the slide closes, thus isolating the pyrometer from the radiated heat.

These measures have resulted in a considerable reduction in the operating cost of temperature control instruments, have reduced the maintenance labor, have improved the conditions of work and provided continuous temperature control on the furnaces.

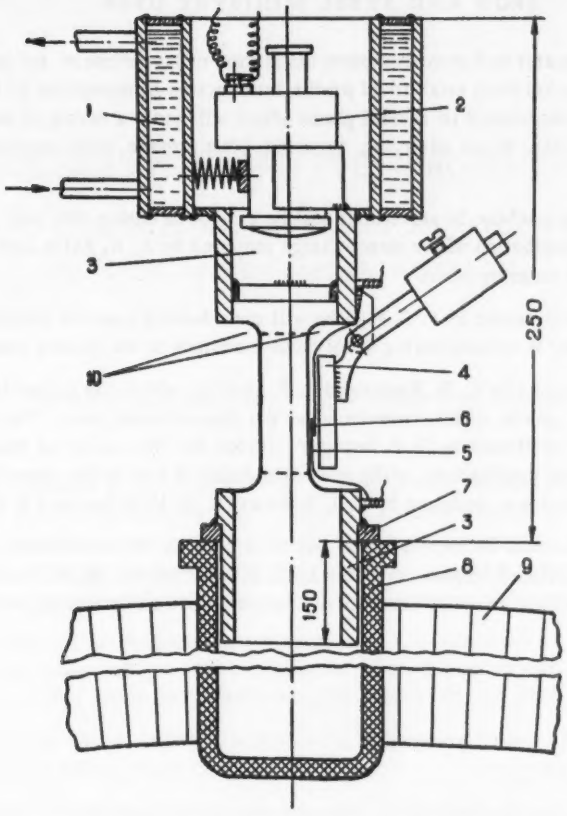

Fig. 5. Diagram of the radiation pyrometer installation for measurement of the arch temperature of a soaking pit.
1) Tuyere with water cooling; 2) radiation pyrometer; 3) sighting tube, diameter 2"; 4) slide; 5) copper wire; 6) soldered joint; 7) support; 8) refractory bowl; 9) soaking pit arch; 10) sight tube cut-aways for ventilation.

Editorial note. When employing radiation pyrometers in automatic arch temperature control circuits, valve changeovers and other devices for protecting the instruments must be provided with electrical contactors for switching in control circuits when the protective slides operate, together with back-up devices for cases of failure.

IRON AND STEEL MINISTRY USSR

In order to achieve a greater and more complete utilization of inventions in the plants of the Ministry of Iron and Steel USSR. a plan has been established for the introduction of inventions in 1957. In the current year technical innovations will be introduced in several plants which will yield a saving of several million rubles and will raise the productivity of labor in ore quarrying, sintering, blast furnace, steel smelting, rolling and other processes.

A. K. Rudkov's sintering machine improvement will be employed during this year on eight Ukrainian sintering strands and a device for the distribution of the sinter charge proposed by A. N. Balon and V. A. Busygin will be introduced at the Vysokogorsk sintering plant.

The mechanical rabbler invented by P. S. Plyuiko will considerably ease the smelter's work on steel alloy furnaces. TheKadievka factory is manufacturing 25 of these machines in the present year.

A new dust-catcher, designed by L. D. Yupko and B. P. Mikhno, which has gained high commendation on the Voroshilov and Makeevka plants will be introduced at the Zaporozhstal plant. The construction of the bottom pouring block will be facilitated by D. P. Bezugly's device and the quality of transformer steel will be improved by employing vacuum installations, while the expenditure of fuel in the open-hearth furnaces will be reduced by introducing gas nozzles as designed by N. I. Kokarev, P. G. Kapichev and S. F. Aleksandrov.

In 1957 several inventions will be introduced at our rolling mills. A considerable saving will be afforded by the automatic stacker for polished tinplate invented by S. A. Usievich and M. N. Smetanin. New methods for removing scale from the surface of rolled sections will be employed and stamping and marking of rolled sections will be mechanized.

ROLLED STEEL AND TUBE PRODUCTION

INFLUENCE OF THE SLAB ROLLING SYSTEM IN THE BLOOMING MILL ON THE QUALITY OF THE METAL SURFACE

L. V. Andreyuk

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The problem of the influence of the rolling system and the amount of squeeze on the quality of the surface of slabs and ingots has not been sufficiently studied. There is a theory according to which ingots should be rolled on the blooming mill with minimum squeeze in the first passes for strict disintegration of the primary structure of the cast metal with frequent manipulation, only subsequently applying the squeeze possible according to the limitations of drag, strength of the rolls and power of the drives.

The Magnitogorsk central factory laboratory has carried out investigations for the purpose of studying the influence of the slab rolling system and the squeeze sequence on the blooming mill with regard to the surface quality of ingots and slabs. Five killed steel smelts and five carbon rimmed steel smelts were rolled.

In order to eliminate the effects of irregularities in the surface properties of the slabs occurring in the soaking pits on the subsequent rolling, each smelt was divided into two halves (odd and even ingots).

One half of the ingots of each smelt was rolled into slabs by the ordinary system of passes with squeezes governed by the power of the drive, and the other half with squeezes reduced in the early passes approximately to half by adding four straight passes.

The system of passes for the 750 x 670 mm killed steel ingots are set out in the table. The rimmed steel ingots were rolled on a similar system.

For the purpose of comparing the surface properties after rolling on one or another system a comparison was made of the labor expenditure (time) for removing the surface defects. No appreciable difference was observed in the labor expended in dressing and consequently in the surface properties of the killed metal rolled with various systems of squeezing in the initial stage of rolling.

In some cases, rolling the rimmed metal with reduced squeezes and with poor quality of the basic ingots even produced a sharp deterioration in the quality of the surface.

There is an opinion according to which manipulating the ingots for the purpose of obtaining a better surface quality for the slabs or blooms should be carried out as frequently as possible.

At the Magnitogorsk works manipulation of the ingots was previously carried out after the first two passes. It was recommended that the slabs should be rolled with manipulation after the first two edging passes and then after each four straight passes.

During the investigations a study was made of the effect of the frequency of manipulation on the surface quality of the metal in rolling killed and rimmed steel ingots.

Three killed low-carbon steel smelts were rolled on the system 2-4-2-4-2-1 and 2-8-2-1 together with six smelts of the same steel on the systems 2-8-2-1 and 8-2-1.

Comparison of the labor expended for removing surface defects in working each half of the experimental smelts indicated that the frequency of manipulation in rolling slabs from the killed steel ingots did not have any effect on the surface quality of the slabs.

Systems for Rolling Killed Steel Ingots

Number of pass	Section, mm	Squeeze Δ, h mm	Extension Δ, b mm	Pass
	a. Rolling to system 2-8-2-2			
1	690×670	60	0	I
2	685×670	5	0	I
	Manipulation			
3	590×695	80	10	I
4	530×705	60	10	I
5	475×715	55	10	I
6	420×725	55	10	I
7	370×735	50	10	I
8	315×745	55	10	I
9	265×755	50	10	I
10	215×765	50	10	I
	Manipulation			
11	660×218	105	3	IV
12	580×220	80	2	IV
	Manipulation			
13	175×600	45	20	I
14	130×610	45	10	I
	b. Rolling to system 2-12-2-2			
1	710×670	40	0	I
2	685×670	25	0	I
	Manipulation			
3	625×690	45	5	I
4	595×695	30	5	I
5	565×700	30	5	I
6	535×705	30	5	I
7	505×710	30	5	I
8	475×715	30	5	I
9	445×720	30	5	I
10	415×725	30	5	I
11	365×735	50	10	I
12	315×745	50	10	I
13	265×755	50	10	I
14	215×765	50	10	I
	Manipulation			
15	660×218	105	3	IV
16	580×220	80	2	IV
	Manipulation			
17	175×600	45	20	I
18	130×610	45	10	I

It appears at first sight that slabs rolled with frequent manipulation have smaller transverse cracks at the small edges than slabs rolled with the smallest number of manipulations. However, dressing of the cracks on one and the other slabs requires equal expenditure of labor. In the one case the area of the cracks is less and the edges are smooth, while in the other case there are funnel-shaped cracks with rough edges. At the same time, there was traced a variation in the cracks of metallurgical origin at the time of hot plastic deformation. From the killed steel ingots, having cracks of considerable length, pieces of metal were cut off with defects by means of the oxyacetylene torch, and then from each piece specimens were taken 100 mm long together with intermediate templets for determining the macrostructure of the initial specimens with a crack in the transverse section.

From each piece of ingot three to five samples were cut and the same number of templets. Each part of the samples was then subjected to hot working with different degrees of shortening, changing the arrangement of the defects on the samples and applying different systems of manipulation.

As indicated by the investigations there was not in any case observed an increase in the depth of propagation of the crack, obtaining through plastic deformation.

In this connection doubt was thrown on the efficacy of manipulation after the first two passes when rolling ingots on the blooming mill.

Preliminary tests were carried out on five smelts of grade 35 and 45 steel. However, the method for assessing the surface quality did not provide sufficient accuracy.

For the purpose of a further test twelve more smelts of grade 08, 10, 15G and 45 killed steel were rolled on a system with manipulation after two and after four passes at commencement of rolling. In this case the ingots were not divided into two groups but were rolled in pairs. One pair of ingots was rolled on one system and the following pair on the second.

After the 720 continuous billet mill the rolled sections were passed also in pairs to the No. 2 and 3 hot shears, the ingots rolled on the one system passing always to the one line of shears. The surface quality was assessed in each case according to the labor expended in dressing the defects in the ingots.

The results of the investigation showed that there is no substantial difference on the basis of labor expenditure between ingots rolled according to the different systems, i.e., the surface quality of the ingots and slabs from killed carbon steel does not depend on the system and sequence of passes.

The surface quality of rimmed metal depends in some cases on the system of rolling on the blooming mill (for example, in rolling ingots of rimmed steel of the old UN-6.6t type into slabs).

Slabs, of the new UNS-7.1t type, intended specially for rolling slabs, and very wide, were rolled with slightly greater squeezes (on average by 60 mm) in the first two edging passes before manipulation. The difference in labor expenditure in rolling these ingots with different frequency of manipulation was on average only 14%.

It was established earlier that the surface quality of slabs of rimmed steel, rolled from slabs of one and the same type and on one and the same system, is better the narrower are the rolled slabs, that is to say the greater the edge squeeze. In order to confirm this, statistical work was carried out on the data relating to the rolling of a large number of slabs from ingots $\frac{730 \times 640}{750 \times 660}$ and $\frac{780 \times 610}{800 \times 630}$.

The surface quality of the slabs was assessed according to the amount of transverse cracks and hollows.

As shown by Fig. 1, the amount of slabs with reject sections is less on narrower slabs and when they are rolled from wider ingots.

A regular relationship was obtained between total edging squeezes and the surface quality of the slabs (Fig. 2).

In order to study the effect of frequency of manipulation on the surface quality when rolling ingots of rimmed steel on blooming mills, six experimental smelts were rolled (08rs steel and 10rs steel): one half of each smelt according to the system 2-2-4-2-2-2-1 and the other according to the system 4-4-2-2-2-1. As demonstrated by a comparison of the labor expended in eliminating defects there was no difference in the surface quality of the ingots obtained.

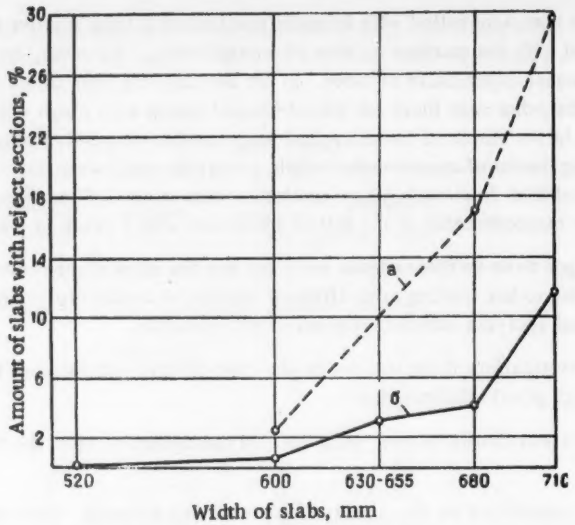

Fig. 1. Influence of the width of rolled slabs on the surface quality.
A) slabs rolled from 750 x 660 mm ingot (7045 specimens);
B) slabs rolled from 800 x 630 mm ingot (20629 specimens).

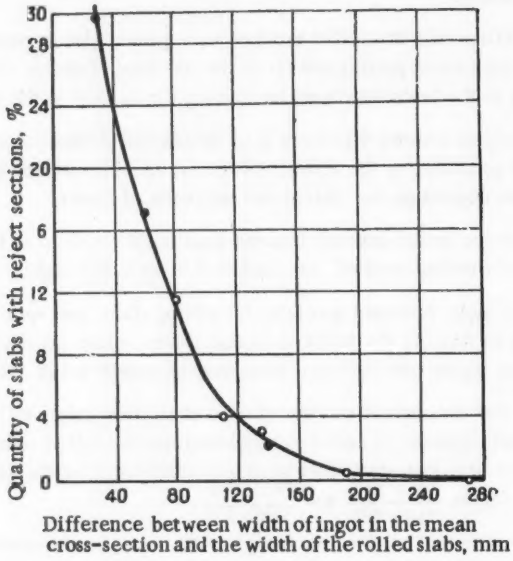

Fig. 2. Relationship between total edging passes and surface quality of slabs.
O) 800 x 630 mm ingot; ●) 750 x 660 mm ingots.

Hence, the surface quality of rimmed metal depends on the system of rolling the slabs on the blooming mill. However, when rolling rimmed steel ingots in four passes without manipulation even in the initial stage the surface quality is not injuriously affected. When rolling slabs the increase in the quantity of surface defects depends on the amount of preliminary squeezing in the perpendicular direction, that is to say on the degree of fusion of the numerous bubbles and the likening of the rimmed steel ingot structure to the structure of the killed steel ingot.

In rolling rimmed steel ingots with a greater number of passes than four without a preliminary edging pass the surface quality of the metal deteriorates to a considerable degree.

Up to 1955 at No. 3 blooming mill at the Magnitogorsk steel works blooms were rolled from ingots of all types according to systems envisaging essential manipulation after the first two passes. Slabs were rolled from killed steel ingots with preliminary edging pass by adding two passes and one manipulation.

After carrying out the present work from the beginning of 1955 carbon steel ingots were rolled according to the system 4-4-2-2-2-1, i.e., with the first manipulation after four passes. By this means it was possible to raise the productivity of the blooming mill when rolling rimmed steel ingots by 2-3% and killed steel ingots 6-7% without injuriously affecting the surface quality of the metal. Slabs were also rolled from killed steel ingots before installing an additional reversing stand with the smallest number of passes for obtaining the given profile and manipulations according to the system 8-2-1. In this way it was possible to raise the productivity of the blooming mill when rolling slabs from killed steel ingots by 14-15% without damaging the surface quality of the metal.

For the purpose of improving the metal surface, slabs should be rolled in advance from rimmed steel ingots with first manipulation after one or two edging passes and then after every four straight passes. In addition, slabs from rimmed steel ingots should be put to rolling in such a way that the initial ingot width exceeds the specified slab width by not less than 100-120 mm.

EXPERIENCE IN THE BETTER UTILIZATION OF ROLLS

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N. U. Paukov, Senior foreman 500 mill

Magnitogorsk Steel Works

The 500 mill consists of nine two-high stands. There is one pass on each stand. The varied batches produced on this mill require an enormous stock of rolls. Together with the systematic improvement in the quality of the rolls and the introduction of high-strength rolls with cast grooves a system of combined dressing of the rolls was employed providing a considerable improvement in their utilization.

For example, the channel iron No. 12-18 roll was designed on the girder method. Since the slope of the outer edges of the flanges of the finished profile of the angle iron is not accounted for in the OST Soviet specifications, the finishing passes are made with a tolerance of 0.5%. Worn rolls were restored only by employing the width tolerance.

In the majority of cases the rolls of the ninth and seventh stands were written off after re-turning once (taking off a metal thickness 20-25 mm) since on rolling with them after subsequent renewal the profile would be outside the tolerance limits in width. The design features of the mill do not provide for the arrangement of channel iron rolling with increased draw-off in the finishing pass and subsequent removal of the opening-up of the flanges on roll-straightening machines.

The following system of dressing the top (open) roll of the finishing stand was adopted at the plant. Two passes are cut in a new roll for rolling channel iron No. 14 with minimum width tolerance. After working both passes (strength of pass 700-800 tons) the roll is re-turned for channel iron No. 16 and further for No. 18. This re-turning does not reduce its strength. In a similar way re-turning is carried out on the closed roll of the seventh stand for rolling the same channel irons. This method of re-turning rolls allows for: double the length of utilization of the rolls of the seventh and ninth stands for rolling No. 14-18 channel iron; re-turning without reducing the diameter of the roll; and an improvement in the quality of straightening of the channel irons through the constant width.

The system of overhauling rolls without reducing the diameter makes it possible to re-turn all the closed rolls of the seventh and ninth stands after wear through rolling channel irons to barrels and to use them again for rolling strip.

For rolling strip use is made of the closed rolls of the eighth stand (control pass), after exceeding the width tolerances subsequent to rolling channel iron. Edging passes are turned on them for the fifth and eighth stands. The preparation in this way of edging passes considerably reduces the amount of lathe work as compared with cutting them on a fresh roll.

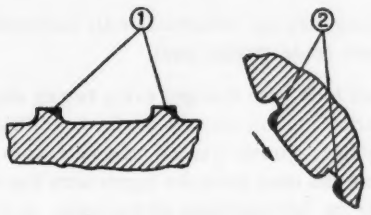
Out of the worn closed rolls of the finishing stand designed for rolling R-18 rails, the open rolls are made for the sixth stand of the same section.

Interest is also provided by the practice of utilizing the rolls for rolling batch sections of reinforcement bars.

The mill rolls No. 50-80 reinforcement round bars on a universal pass, attaining a high productivity. The durability of all the passes with the exception of the finishing pass provides for the rolling of very large batches of metal. The mean strength of the roll of the finishing stand is 650-700 tons.

Attempts to increase the strength of the passes by using harder rolls resulted in a higher level of reject rolls in turning the grooves for the spiral projections.

In the majority of cases the cause for changing the pass was the inadequate depth of the spiral projections on account of wear of the roll (see diagram). In this case there is wear on one side of the milled groove of the roll. Wear occurs on the edge first coming into contact during rotation of the roll and, as it were, impeding egress of the metal from the seat of deformation. The degree of wear at the opposite edges is negligible. This is explained by the sharply developed lead of the metal which amounted on average for No. 80 rounds to approximately 14%, No. 70-12% No. 60-10% and No. 50 approximately 8%. After working all the passes the rolls of the finishing stand were changed over, that is to say the roll occupying the top position in the previous campaign was replaced by the bottom roll and vice versa. The results were completely satisfactory since in operation the rolls were employing the edges of the milled grooves which were not worn.



Incomplete projections (1) and wear of the finishing pass groove (2)

In this way without re-turning and milling the rolls it was possible to roll in each pass double the quantity of reinforcement steel of batch section No. 50-80.

In order to reduce the amount of metal taken off the diameter for this group of sections the worn rolls for rolling batch sections were turned in the following manner: from passes No. 50 No. 60 were made, from No. 60- No. 70, and so on.

Experience has shown that combined re-turning of rolls provides a considerable increase in the degree of utilization of the rolls.

IMPROVED OPERATION OF SHEET MILL ROLLS ON THE ROUGHING STAND

S. F. Kumachev - Senior foreman, 2300 mill

I. D. Dmitriev - Chief research engineer, central factory laboratory rolling group

Stalin steel plant

The three-high roughing stand of the two-stand 2300 sheet rolling mill has cylindrical rolls diameters 750/570/750 mm with barrel 2300 mm long. The rolls are made from grade 60KhG steel.

From the moment of starting the mill (1953) the top and bottom rolls were changed every twenty days, while the middle roll was usually changed after every ten days operation. In this way the set for re-rolling the three-high stand comprised two outer and two middle rolls. Investigations showed that the rolls were worn disuniformly along the length of the barrel. The left end of the roll wore the most since the ingots were conveyed along the left side of the approaching roll train and were delivered to the left-hand side of the stand. In 1955 guides were set up on the approach roll train which centered the ingots before delivering them to the stand. The rolls then wore much more uniformly and to a smaller degree (Fig. 1).

Since 1956 the system of working the rolls of the roughing stand has also been changed for the purpose of increasing the productivity of the mill. All the rolls are now changed daily and the set of rolls for the three-high stand comprises two outer rolls and one middle roll. Wear of the rolls has been considerably reduced (Fig. 2) while the conditions of operation of the mill have been improved. The rolled section now displays a smaller difference in thickness than previously. This considerably facilitates adjustment and operation of the finishing stand, improves the condition of operation of the spindles, bearing, limiting chocks and other equipment of the three-high stand.

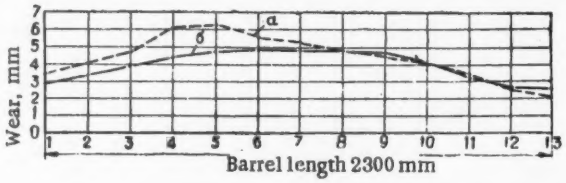


Fig. 1. Wear on the outer rolls of the three-high stand. a) Before installing the guides; b) after installing the guides

The overhaul of the rolls was also reorganized. While previously after operation the rolls were re-turned on the roll-turning lathe and only the finishing rolls (iron) were polished on the roll polishing lathe, since 1955, all the rolls of the four-high and three-high stands were polished without preliminary re-turning on the roll turning lathe. This was possible by using together with the usual carborundum wheels polishing wheels with bakelite binder.

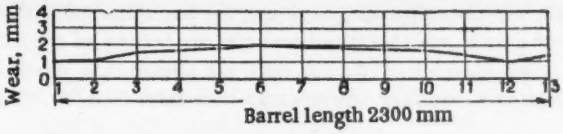



Fig. 2. Wear on the outer rolls of the three-high stand after changing the system of operation.

By adopting these measures it was possible to considerably raise the service life of the rolls, the expenditure on rolls being reduced as compared with 1955 from 0.426 to 0.306 kg/t of rolled section, that is to say by 28%.

LENGTHENING THE TAKE-UP ROLL OF A BLOOMING MILL

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V. I. Khvatov – Design department, Magnitogorsk works

At the Magnitogorsk works further increase in the production of the blooming mills after reconstructing some units and installing additional stands was held back by the irregularity of delivery of the hot ingots to the mills, particularly from the more remote soaking pits.

The roller beds to the blooming mills in the soaking pit sections were arranged in such a way as to be served only by the first crane. With the slightest irregularity of the ingot cars considerable delay was introduced in delivering the ingots to the roller bed.

For the purpose of obtaining uniform delivery of the hot ingots it was decided to lengthen the roller beds of both blooming mills approximately by 30 m (three sections of 11-15 rollers each), that is to say almost up to the beginning of the third group of soaking pits (Fig. 1).

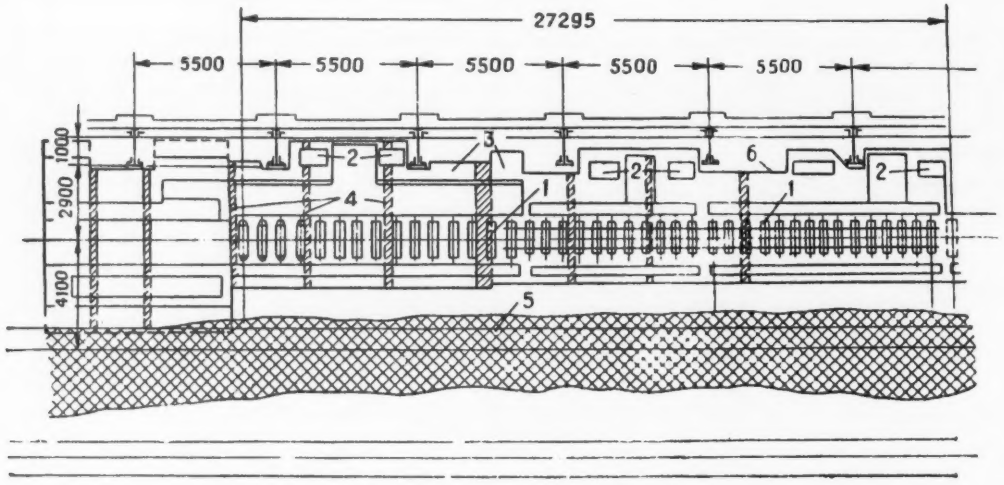


Fig. 1. The extended roller bed:
1) Sections of the extended roller bed; 2) drives for the additional sections; 3) reinforced-concrete foundation blocks; 4) joints between blocks, filled with concrete; 5) position of temporary foundation pit; 6) contours of the reconstructed section of the existing foundation.

The extension to the roller beds reduced the travel of the ingot car and increased the rate of delivery of the hot ingots from the far soaking pits approximately by 20% while if the ingot car was out of action the ingots could be brought up by means of the grab cranes. The roller bed was extended in such a way as to be served by two or three cranes.

A difficulty in extending the roller bed and transferring the tilter with the plant operating was that all the construction and erection had to be carried out in a very restricted time, during the three-day overhaul of the blooming mill. It was necessary to dismantle the tilter and part of the ingot track, excavate more than 500 m^3 of earth, place more than 270 m^3 fresh reinforced-concrete and to assemble approximately 300 tons of equipment.

In order to successfully carry out the work it was decided:

1) to construct the foundations under the roller bed and tilter from previously prepared large reinforced-concrete blocks of a volume of 30 m^3 and weight 75 tons;

2) to assemble the blocks and the large prefabricated sections of the roller bed by a special device with a hoisting capacity of 75 tons, set up on the metal work of the soaking pit crane;

3) before commencing the planned overhaul to previously cut a foundation trench alongside the future block foundation of the roller bed.

The depth of the foundation was set at 2.5 m from the ground level. The reduced depth as compared with the depth of the foundation of the operating roller bed was connected with the fact that the fresh foundation was erected in more solidly consolidated ground as a result of the operation of the ingot car. This considerably reduced the volume of earth and concrete work.

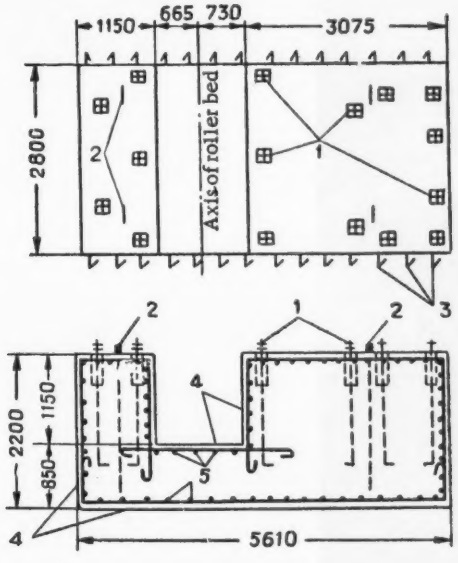


Fig. 2. Reinforced-concrete block:
1) Anchor bolts; 2) section eye-pieces; 3) projecting ends of reinforcement, at block junction; 4) alternating section reinforcement, 16 mm diameter; 5) 10 mm diameter reinforcing bars.

The foundation under the tilter and roller bed extension consisted of nine large prefabricated reinforced concrete blocks (grade 150 concrete) with a volume of 23 to 30.5 m^3, 2-3.3 m long, 4.8 to 7.5 m wide and 2.5 m thick (Fig. 2).

The box section of the blocks provided a channel under the roller bed for collecting the scale. The blocks were reinforced throughout the profile with 10 mm diameter steel rods every 200 mm and 16 mm diameter alternating sections at 120 mm pitch. In order to obtain a better joint, projections were provided at the faces of each block to 150 mm every 400 mm. The anchor bolts were sealed in with 150 × 150 mm insets.

The blocks were concreted in the construction repair shop from a special platform set up along the rail track and loaded onto 120-ton open-hearth furnace wagons for moulds with two steam track cranes of 45-ton capacity.

The foundation blocks and prefrabricated roller beds were assembled by means of a special device with a hoisting capacity of 75 tons, set up in a fixed position on the metal structure of the soaking pit crane bridge. The gear (Fig. 3) was arranged above the axis of the ingot car track and consisted of a 10 ton erecting crane, a frame with five pulley blocks, a hanger with six blocks and hauling cables with eyeholes. During assembly the foundation blocks and gear were moved along the soaking-pit bay by means of the crane travelling gear, and across by means of a cable fastened to the hook of a small winch mounted in the crane cabin.

The drive and crane travel mechanism were controlled from the operator's cabin.

The erection and assembly work for extending the roller bed were carried out by the continuous method. Immediately after dismantling the existing equipment and after completing the earth-work the foundation was prepared. As the foundation became ready the foundation blocks and gear were assembled. As soon as the ingot car was stopped the earth was driven into the foundation pit previously dug out by excavator alongside the roller bed. A layer of rubble 150 mm thick was placed under the foundation blocks.

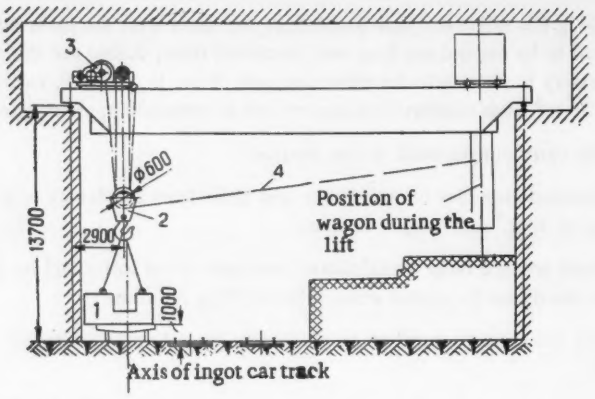
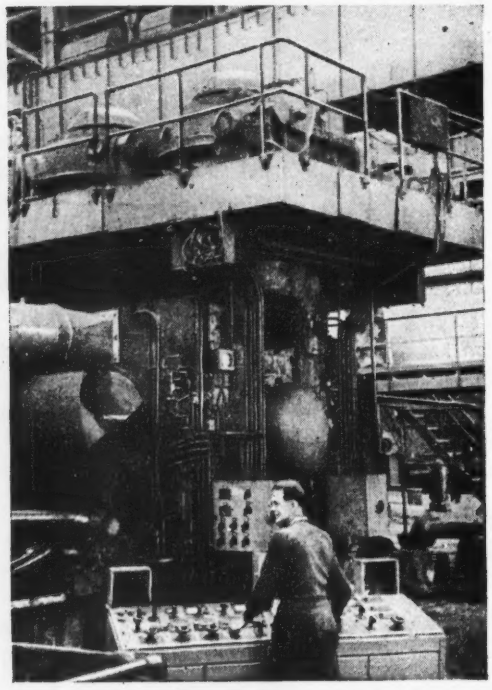


Fig. 3. Erection gear:
1) Foundation block; 2) 75 ton six-block tackle; 3) 10 ton winch; 4) hauling cable.



A 1200 reversing mill was commissioned in August 1956 at the Zaporozhstal cold rolling plant, manufactured at the Ural heavy machine construction plant. The design strip rolling speed on this unit is 2.5 times greater than on existing mills. The 1200 mill is shown in the photograph.

After setting up the first pair of foundation blocks a start was made immediately on erecting the roller beds. As soon as the first section of roller bed was checked a start was made on filling and concreting the joints between the blocks using concrete on alumina cement.

After completing dismantling of the ingot tilter the old reinforced-concrete was dug out at the junction of the roller bed extension and the existing roller bed.

During the period of the three-day overhaul on the mill all the operations involved in extending the roller bed were carried out simultaneously.

In this way it was possible to deliver the hot ingots from the soaking pits to the blooming mill without interruption.

Hence, the method of work organization adopted for extending the roller bed proved completely effective and can be confidently applied at other plants.

CENTRIFUGAL CASTING OF 50mm CAST IRON STANDPIPES

Cand. Tech. Sci. B. D. Khakhalin and Engineer A. N. Smolyakov

Centrifugal casting is used for the production of parts of machines, mechanisms, structural equipment and household articles. Recently, centrifugal casting has also been used for the production of pipes.

Workers of the All-Union Scientific Research Institute for Tube Production, together with the Makeev Pipe Foundry have developed a centrifugal process of casting 50 mm cast iron water pipes.

The stationary process of casting 50 mm pipes suffers from a number of serious drawbacks: need for heavy manual labor, production of large amounts of dust and gas in the foundry, low labor productivity, poor quality of the products, and high metal consumption. All these disadvantages are, to a large extent, eliminated by centrifugal casting methods.

To cast the metal in a chill mold by centrifugal casting, it is necessary to have a strong, long mold (about 3.5 m or more long) of very complicated construction. In addition, the phosphorus content of the metal has to be increased, so that the latter will have the necessary fluidity to flow along such a long and narrow mold. Increase in the phosphorus content of the metal, however, results in a marked reduction in the mechanical properties of thin-walled, small-diameter pipes.

Following a series of investigations into different methods for centrifugal casting of specimens of cast-iron pipes, a method of casting in sand-lined shells was selected. This method has a number of very important advantages:

1) Minimum metal consumption due to low quantity of pipe rejects and minimum loss of metal during casting;

2) High mechanical properties of the pipes, absence of surface chill, and elimination of the need for subsequent heat-treatment – an important point in the case of ordinary gray iron and alloy cast iron;

3) Low consumption of mold mixture, due to small overall dimensions of the pipes, and the elimination of expensive chills, subject to a high rate of wear;

4) Comparatively simple construction of the centrifugal machine, affording the possibility of fully automatic operation.

The disadvantages of centrifugal pipe casting compared with casting in metal chill molds are: necessity for the installation of accessory machines and plant for ramming and knocking out the molds and for cleaning sand crust off the outer surface of the pipes, variation in thickness of pipe walls if the mold is incorrectly rammed and reduction in the output of the machine due to slow solidification of the pipe walls.

The centrifugal machine TsM – 50/4 installed at the Makeev Pipe Foundry for casting 50 mm pipes is shown in Fig. 1.

For preparing the molds, i.e. ramming the lining, an already existing vertical mold-ramming machine was used at the Makeev foundry. The technical process of centrifugal pipe casting is as follows:

An empty mold shell and fresh molding mixture are supplied to the molding machine. A pattern of the pipe is placed in the vertical shell and is rigidly centered at the bottom by the pattern of the pipe socket and at the top by a special rotating centering device. If the pattern is not mounted exactly co-axially with the shell or if it is displaced from its original position during the making of the mold, the wall thickness of the resulting pipe will vary. After mounting the pattern in the shell, the mold mixture is fed through the top centering device of the molding machine rammer. The rammers move up and down with simultaneous rotation round the pattern during the operation of the machine and compact the mold mixture fed into the shell. When the

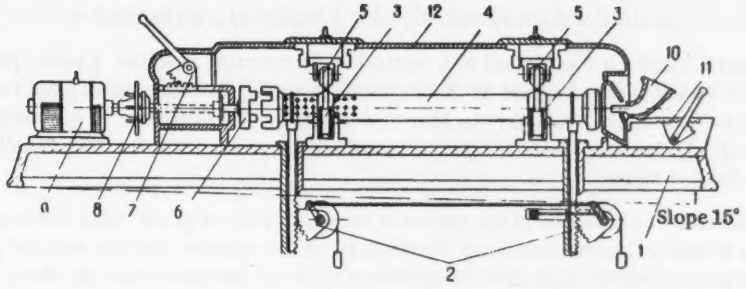


Fig. 1. Diagram of constant slope centrifugal machine. 1) frame; 2) lifting mechanism for lifting and rolling out previously cast mold shells and receiving new mold shells; 3) supporting rollers on which the mold shell rotates; 4) shell; 5) upper pressure rollers; 6) shell-driving clutch; 7) clutch-operating mechanism; 8) coupling; 9) electric motor; 10) pouring funnel; 11) funnel in-feed mechanism; 12) machine casing.

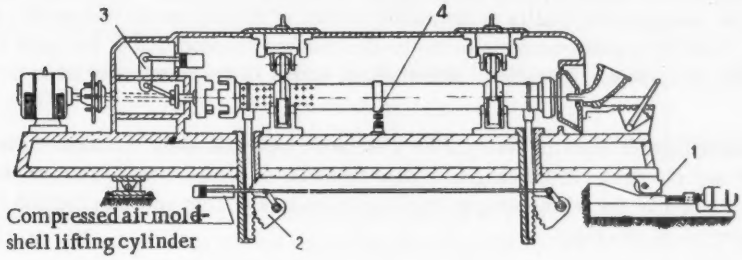


Fig. 2. Diagram of centrifugal machine with ramp device.

shells are full, the rammers are retracted fully upwardly, the centering device is removed and the molding machine is moved aside. The pattern is removed by means of an overhead crane, the surface of the mold is coated with a graphite-coke wash and the mold is set over a burner for drying. The dried mold is then turned into the horizontal position, its ends are closed at one end by the socket core and at the other end by a stop ring. The mold thus prepared is put on the receiving stand of the centrifugal machine.

In continuous operation the mold is brought automatically to the machine and set in rotation, and a measured quantity of molten iron is poured in by hand. To enable the molten iron to spread rapidly along the entire mold, the machine is inclined at an angle of 1.5° to the surface of the floor. With such inclination, a slight permissible longitudinal variation in wall thickness of the pipe occurs during solidification of the metal. When the metal has solidified, the mold is ejected from the machine on to the stand for cast molds, is allowed to cool and is then moved into a horizontal stripping chamber. The sand layer of the mold is broken up by pushing through it a pipe nozzle fed with compressed air. The mold mixture is blown out and the pipe nozzle readily bores a hole through the mixture. The cast pipe is then completely removed and is sent to the cleaning and inspection shop. The mold shell is cleaned from residues of burnt mold mixture and is ready for re-ramming.

The experience gained at the Makeev Pipe Foundry in making 50 mm pipes by centrifugal casting has been utilized at the Pervoural Starotrubny works, where the workers introduced their own modifications into the casting technique and construction of the machine.

The centrifugal machine was provided with a ramp-lifting mechanism 1 (Fig. 2), which tilts the machine

slightly during casting and returns it to the horizontal position during solidification of the metal. This removes any variation in wall thickness of the pipe. The lifting mechanism 2 has no weights. This means that the mold shells have to be raised by hand, an undesirable feature. The clutch 3 is operated by compressed air. To reduce the period of unproductive operation of the machine, the shell is braked by a brake block 4.

The important modification is that instead of a vertical mold ramming machine, a worm-plunger machine is used, the construction of which was designed by the engineers Semenov, Drozdov and Bogorad (workers in the Design Department of the Ministry of Transport Machine Construction). The advantage of this method is that ramming the mold shell is horizontal, i.e. in the position in which the molds are cast. This simplifies transport of the shells and finished molds.

An important technical modification of the process is the application of green- sand molds with a thin layer of mold mixture instead of black-washed and dried molds. It was assumed that the result of such a modification would be a considerable increase in the productivity of the machine, since the more intense cooling would reduce the solidification period of the metal in the mold. In actual fact, reducing the layer thickness of mold mixture results in a reduction in the yield or give of the molds and makes shrinkage more difficult, causing frequent fracture of the socket part of the pipe. The difficulty of shrinkage is still further increased by the presence of a chill in the socket part. In addition, reduction in the layer of mold mixture results in rapid heating of the mold shell, which gives rise to the risk of distortion and the need for artificial cooling.

Rejects of pipes cast at the Pervoural Starotrubny works exceeds the permissible standards. Some of this waste is due to the fact that the technique has not yet been adequately acquired. The remainder of the wastage is due to the fact that the modifications introduced into the technique are not perfect. The worm-plunger machine does not yet ensure proper quality of the molds because of the inaccurate movement of the plunger and its rapid wear. Casting in green-sand molds results in a marked deterioration of the outer surface of the pipes, due to increase in sandcrust formation. Nevertheless, output in the Starotrubny works is higher than at the Makeev works.

The All-Union Scientific Research Institute for Tube and Pipe Production, the State Planning Department for Steel Production and the production managers of pipe foundries of the Ministry of Ferrous Metallurgy of the U.S.S.R. ought to improve the worm-plunger machine or produce a more accurate method of making molds for centrifugal casting of 50 mm pipes.

SCREENING AND VENTILATION OF ROLLING MILL CONTROL CABINS

Cand. Tech. Sci. Ya. A. Shtromberg and A. F. Babalov

All-Union Scientific Research Institute for the Protection of Labor of the

S. M. Kirov All-Union Central Council of Trade Unions

In rolling mills, the control cabins are often located in positions of high thermal radiation. At the Zakavkaz Metallurgical Works, the control cabin of the billet stock of the blooming mills (Control Cabin 5), where two operators usually work, is most uncomfortable in this respect. The cabin has hot steel on three sides of it and is exposed to intense thermal radiation: the temperature of the outer surface of the cabin walls reaches 47°C and that of the floor 40-60°C. The radiation of heat from the hot walls and floor of the cabin affects the operators. In summer, the air temperature of the cabin reaches 45-50°C.

The temperature of the outer surface of the glazed part of the walls is 53-54°C and that of the metal part is 62°C.

In an endeavor to reduce the air temperature and radiation intensity, a system of ventilation and screening was devised for the control cabin, comprising screening structures for stopping the flow of radiant energy from the incandescent metal and special ventilation devices for cooling the incoming air (Fig. 1).

The control cabin walls are screened by means of aluminum foil placed at a distance of 20 mm from the outer surface of the walls.

The glazing of the cabin is triple (the first pane is stalinite), with an air gap between the panes of 15-20 mm.

The second and third panes are in detachable frames inserted in grooves in the main frames of the cabin. The entire operation of removing the panes and replaci ng them by spare panes does not take more than 1-1 $^1/_2$ minutes.

The panes have to be removed for cleaning once every 3 or 4 shifts.

Such an arrangement reduces the radiation intensity at the working place by 3-5 times. For example, if the surface temperature of the first pane of the glazing reaches 70°C or more, it does not exceed 35-38°C on the third set of panes.

The effect of the triple glazing is still further improved by producing an artificial current of air between the panes.

The principle of spraying water into a moving current of air is used for cooling the air. According to calculations, the excess of heat in control cabin 5 after screening is 2000-2800 cal/hour, depending upon the working conditions and the time of the year.

Experience has shown that the amount of air supplied for one operator should be 1500 m^3/hour, i.e. for two people it should be 3000 m^3/hour, which is about 3500 kg/hour.

The ventilation of the control cabin is carried out with 90% recirculation of the air. Figure 2 shows the general view of the fan chamber of the cabin.

In the hottest period, when the temperature in the rolling mills is above 30-32°C and the relative humidity is higher than 35-30%, the air is cooled by means of mains water with a consumption of 10 liters/min. For the rest of the time, it is sufficient to cool the air by recirculation water, which does not necessitate consumption of water. In this case, the air is not recirculated and the plant is always supplied with air from the rolling mills.

Cooled air is supplied to the upper part of the cabin through special nozzles for directing the stream along the ceiling or on to the working place. The direction of the air stream is varied by the operators by turning a handle in the working place.

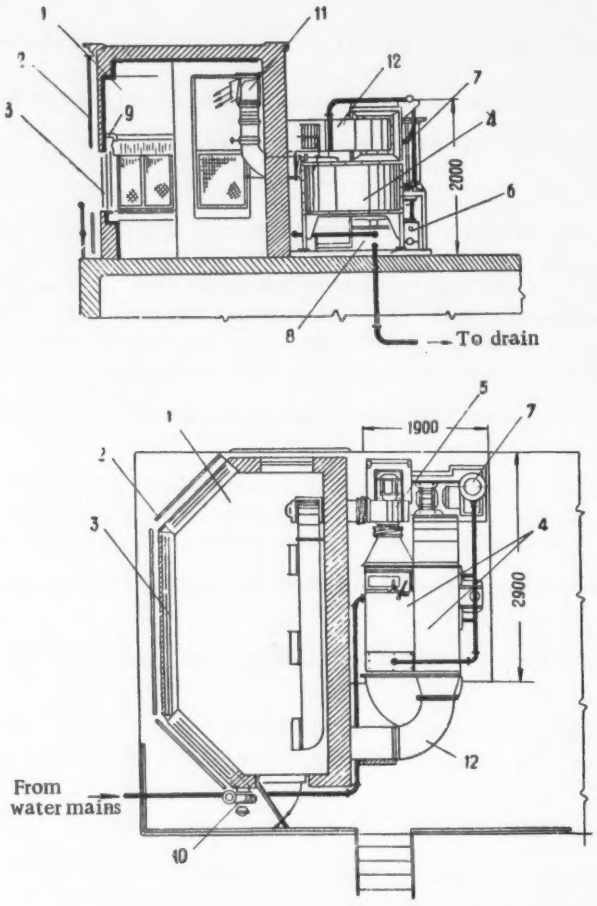


Fig. 1. Installation for ventilating and screening control cabins. 1) cabin; 2) screens; 3) triple glazing; 4) air cooler; 5) cabin fan; 6) pump; 7) filter; 8) water tank; 9) exhaust from triple glazing; 10) glazing fan; 11) air inlet duct; 12) recirculation air duct.

For the winter working period, the provision of a heater is contemplated for heating the air supply to a temperature of 17-18°C. The ventilating plant permits the temperature in the cabin to be maintained at 27-29°C for a mill shop temperature of 36-41°C and an outside temperature of 32-35°C. Usually, for the above-mentioned outside temperatures, the temperature in the cabin has been 45-48°C.

Following trials, the installation has been in operation at control cabin No. 5 of the Zakavkaz Metallurgical Works and is giving good results. The cost of such an installation is 14-15 thousand roubles, while air-conditioning plant for one cabin costs up to 150 thousand roubles, excluding heavy costs for erection and maintenance.

The new installation is also a good one because it can be made without trouble at any works in any machine workshop and can be used in any given conditions of either climate or size, while installations with

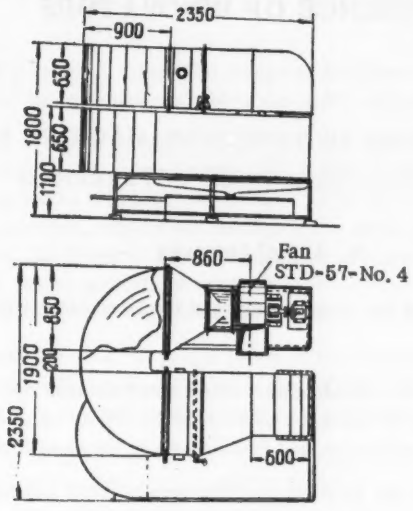


Fig. 2. General view of the fan chamber of the cabin.

refrigerators and automatic control have to be manufactured centrally, and consequently it is impossible to allow for all the peculiarities obtaining at each works.

At the present time, the Ministry of Ferrous Metallurgy of the USSR has invited 18 of the largest metallurgical works to equip rolling mill control cabins with the above-mentioned installation.

EXPERIENCE OF INNOVATORS

WORKING METHODS OF SINTERING-MACHINE FOREMEN A. M. BUZINER AND M. S. PETRENKO

V. Lovchinovsky

(Deputy Chief of the Sintering Plant of the Dzerzhinsk Works)

The quality of sinter determines to a considerable extent the successful operation of the blast furnaces. Sinter should be easily reducible, lumpy in character and mechanically strong, should contain self-fluxing gangue and be low in impurity and should be porous. Fluctuations in the chemical composition of the sinter should be small.

The most important condition for the production of high-grade sinter and for the uniform and highly productive operation of sintering machines is the continuous supply of the stock bins with well-bedded raw materials, charging of the latter bin by bin and proportioning of the charge according to the results of chemical analysis. The granulometric composition of the charge, i.e. its percentage content of particles or grains of different size is also of considerable importance.

In the event of modifications in the proportioning of the various components of the charge or a reduction in their quality, the sintering machine operators take immediate steps to re-proportion the charge and to establish new sintering conditions, so that fewer mistakes are made in conducting the technical sintering process and high-grade sinter is produced.

For eight months in 1956, the crew of A. M. Buziner and M. S. Petrenko had a yield of second-grade sinter 175 tons less than the first crew and 2526 tons less than the third crew.

Preparation of the charge for sintering. The difficulty in sintering fluxed agglomerate of high basicity depends to a large extent on the quality of the lime used for fluxing.

Krivorog ores are noted for their relatively high silica content, and therefore a considerable quantity of lime is used in the charge for fluxing them, so that the bulk weight of charge and sinter cake is diminished.

A. M. Buziner and M. S. Petrenko keep a close watch on the quality of the crushed material and the chemical composition of the coke breeze (not more than 15% ash and not less than 90% of the 0-3 mm fraction after crushing). Normal crushing of the coke breeze ensures the necessary conditions for intense combustion and uniform development of temperature in the sintered cake and provides the possibility of normal sintering of the charge. This is also achieved by accumulating the large lumps of the charge on the grates of the sintering machine. With coarse crushing of the coke breeze, its content is increased in the lower part of the charge bed, so that there is excessive production of liquid phase on the grates, resulting in a reduction in sintering intensity and the sinter is burnt on to the pallet grates.

The sintering-machine foremen pay particular attention to early adjustment of the moisture content of the charge. The moisture content of the charge is determined every hour. A. M. Buziner and M. S. Petrenko have reached a point at which the variation in moisture content is not more than 0.5% in most cases.

Due to the use of excessively moist ores and returns, the moisture content of the charge may be above the optimum value; in such a case, the sintering-machine foremen usually try to increase the ignition temperature of the charge. If this is insufficient, the rate of movement of the sintering plant is reduced and the carbon content of the charge is increased somewhat. This ensures regular operation of the sintering process resulting in the production of high-grade sinter.

The carbon content of the charge is controlled in accordance with the glow on the surface of the sinter cake as the pallets leave the burners, and also in accordance with the appearance of the sinter cake fracture as it leaves the strand.

Loading the charge on to the pallets. When the charge is distributed correctly, it lies evenly against the sides of the pallets and against the machine grate over the width and length of the entire strand.

Incorrect distribution of the charge means that the air is drawn through it unevenly over the width of the pallets; the quantity of air passing through increases from the middle of the pallets to the sides, resulting in inadequate sintering of the charge in the middle of the pallets, while at the sides the sinter will be dark. This reduces the productivity of the machines, impairs the quality of the sinter and has a negative effect on the hydraulic consolidating operation. At the same time, the quantity of returned fines is increased and its quality impaired, which in its turn has a detrimental effect not only on the sintering process, but also on the working conditions of personnel in the returned sinter quenching plant, due to excessive dust formation.

The sintering machine foremen check the proper stroke of the swinging feeder, and if necessary regulate it so that there is a larger quantity of charge at the sides of the pallets; this ensures uniform gas-permeability of the charge over the entire width of the pallets and length of the sintering machine.

If it is necessary to alter the height of the gate along the length of the rotating feeder they select a more rational profile of the lower edge of the slide on the rotating feeder.

Another essential condition for satisfactory conduct of the sintering process is the constancy of level of the charge on the deflector plate. Observation of this condition ensures constant increase in density of the charge along the sintering strand, creating the conditions for normal sintering of the charge on the pallets. A. M. Buziner and M. S. Petrenko therefore watch the operation of the swinging feed and if necessary adjust its stroke. The angle of inclination of the deflecting plate as a rule is rarely altered. It is adjusted in accordance with the coarseness of the charge and the need for separating the bed by natural segregation.

The sintering machine operators watch the level of the charge in the hopper above the rotating feeder, keeping it to a constant minimum, so as to eliminate the possibility of excessive compacting of the charge and consequent reduction in its gas-permeability, when its level in the hopper is high. In addition, fluctuations in the level of the charge in the hopper and its irregular charging by the swinging feed result in segregation by size and reduced quality of the sinter.

The surface of the charge is kept horizontal and smooth. The leveller behind the deflecting plate is always in good condition.

The sintering machine operators do not allow harmful blowholes to appear, systematically ensure proper conditions of the grates and stop up the blowholes in good time by means of fine sinter.

Ignition of the Charge and the Sintering Process

The charge is ignited by the igniter using blast-furnace gas. The sintering machine operators carefully watch the condition of the charge on the sintering strand and its ignition temperature, making constant use of the control instrument readings for charge temperature, igniter flame temperature, speed of the sintering machine, gas consumption, gas pressure, air consumption, flue gas temperature and vacuum in the windboxes.

For normal ignition of the charge using blast-furnace gas, the flame temperature should be 900-1170°C. For this, the gas/air proportion should be 1.05-1.10 m^3 of air to 1.0 m^3 of gas. For uniform, intense ignition of the charge, the moisture content of the latter is also of considerable importance. The sintering machine operators do not allow the burner flame temperature to become too high, in order to prevent fusion of the top layer of charge, which would reduce the gas-permeability and slow up the sintering process. Nor do they allow the temperature of the charge-igniting flame to become too low, since this results in reduced mechanical strength of the sinter. A drop in temperature is indicated by a reddish color of the flame. The sintering process is then sluggish and uneven and the yield of useful sinter is reduced. If there is insufficient gas for igniting the charge and if the charge is very wet, the sintering strand is not allowed to travel at a high speed, or the height of the charge bed is increased. Complete and constant sintering of the charge down to the grates is one of the most important conditions of normal operation of the technical process. To this end, the sintering machine

operators systematically watch the waste gas temperature in the last two windboxes, this being an indication of the sintering process throughout the depth of the cake, and serves as a most reliable and accurate criterion for determing the complete sintering of the charge down to the grates.

The temperature of the waste gases in the twelfth and thirteenth windboxes is kept within the limits 200-300°C, and is not allowed to go above that, since a high temperature of the waste gases in these windboxes indicates an excess of carbon in the charge, while a low temperature indicates a deficiency of carbon. The sintering machine speed is maintained at such a rate that the waste gas temperature in the twelfth windbox is 20-50°C higher than in the thirteenth windbox; there is then complete sintering of the charge down to the grates. In addition, the carbon content of the returns which should be very low, not exceeding 0.8% is also used as a reliable indication of the completion of sintering of the charge down to the grates.

Air conditions. The air conditions have a substantial influence on the sintering process.

A. M. Buziner and M. S. Petrenko control the air conditions in accordance with instrument readings and require the workman in charge of the suction fans to see that the slides of the latter are kept fully open, since only then is the power of the fans fully utilized. In addition, they watch the vacuum in front of the suction fan.

The vacuum in the first and second windboxes is kept within the limits 600-800 mm water gage and is regulated by valves; from the third to the tenth windboxes inclusive, the vacuum is kept within the limits 1000-1100 mm water gage, in the eleventh the vacuum is 1000 mm water gage, in the twelfth 900 mm water gage and in the thirteenth 800 mm water gage. The vacuum must be watched, since an increase in the vacuum means a reduction in the gas-permeability of the charge.

An increased vacuum may be due to excessive carbon and moisture content of the charge, insufficient moisture in the charge, reduction in particle size of the charge, excessive consolidation of the materials when charged on to the sinter strand, excessive thickness of charge bed, high speed of sintering machine strand and fusion of the top layer of the charge under the igniter. The vacuum may be diminished because of increase in detrimental blowholes in the bed, abnormal increase in the particle size of the charge or inadequate speed of the sintering strand pallets, for which sintering is completed before the penultimate vacuum chamber.

Stubborn and progressive diminution in the vacuum, especially in the third and fourth windboxes, indicates that the consolidation by the sintering machines has deteriorated, or there has been progressive fouling of the multicyclone elements or a reduction in the free cross-section of the gas discharge system, especially in the outlets from the windboxes.

A. M. Buziner and M. S. Petrenko therefore always see that the vacuum conditions from the third to the fourth windbox are normal, and that the vacuum in front of the suction fan is slightly greater than in the above-mentioned section of the sintering machine.

EXPERIENCE OF NEW TECHNIQUES

ORGANIZATION OF ROLL CHANGING AT MILL 650

L. I. Boldyrev

Mill Foreman of Heavy Section Shop

K. S. Gudovshchikov

Engineer-Investigator Central Works Laboratory, "Azovstal" Works

One of the principal ways of increasing future output of rolled products is to reduce the unproductive stoppages due to roll changing and adjustments.

Reduction in the time required for roll changing and adjustments at mill 650 was made possible by the advanced working experience of the crew led by Foreman P. D. Krishtofovich.

The finishing train of mill 650 consists of two three-high stands and one two-high stand situated in line. The maximum diameter of the working rolls of all the stands is 680 mm, the length of the working roll body of the three-high stand is 1700 mm and that of the two-high stand is 1200 mm; maximum motor speed for the three-high stand is 180 r.p.m. and for the two-high stand it is 220 r.p.m. The rolls of the two-high working stand are driven by a 1500 h.p. electric motor having speeds of 0 – 100 – 220 r.p.m.

The mill is equipped for rolling rails R – 18 and R – 24, beams Nos. 16 – 22, sleepers Nos. 16 – 22, square sections from 60 to 120 mm, rail fixtures for rails R-50 and R-65, heavy angle sections – equal sided 200 x 200 and 150 x 150 mm and unequal sided 120 x 200 mm, mine support sections and other sections rolled from the products of blooming mill 800.

Roll changing at mill 650 is carried out in the intervals between rolling one section and rolling another section, and also when the roll passes are being machined, in which case roll changing sometimes has to be carried out on one, two or even three stands at the same time.

During roll changing, Foreman P. D. Krishtofovich pays special attention to the proper preparation of stands and rolls, correct disposition of workmen and operators, rational use of electric cranes and proper sequence of the operations, since upon this depends the speed and quality of the work done.

Preparation for roll changing. Preparations are usually commenced in good time by the crew of roll setters and the equipment shop. Preparation is under the direction of the senior mill foreman who indicates the rolls and equipment for changing and organizes their delivery to the necessary positions (Fig. 1). The mill foreman first of all inspects the rolls as received from the roll turning shop; dirt and grease are removed from the roll necks to reveal any flaws on them (scoring, pitting, etc).

The sets of rolls for stands I and II are selected so that the difference in their diameters for the two stands does not exceed 30 mm. Considerable differences in the roll diameters may lead to damage during rolling, since the spindles are then working with considerable skew, the top and bottom rolls of stand I work without clearance in the middle and this in its turn will result in fracture of coupling boxes, spindles, wobblers and roll necks.

When the rolls have been inspected on the outside, their diameters and the thickness of the textolite bearing on the bottom chock of the middle roll are measured. These dimensions have to be known so that the position of bottom roll and brackets (hangers) can be rapidly adjusted during assembly of stands and spindles, before the coupling boxes are placed in position.

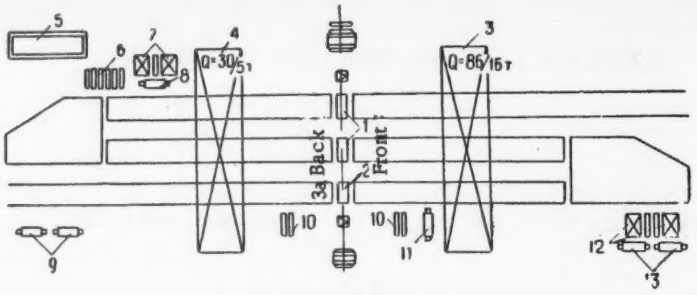


Fig. 1. Diagram showing arrangement of plant, rolls and equipment in preparation for roll changing. 1) three high stands; 2) two-high stand; 3) No. 1 crane; 4) No. 2 crane; 5) spare stands; 6) equipment for stands I and II; 7) pyramids for assembling housing caps and top roll of two-high stand; 8) top roll of two-high stand; 9) middle and bottom rolls of three-high stands; 10) equipment of two-high stand; 11) bottom rolls of two-high stand; 12) pyramid for assembling housing caps and top rolls of three-high stands; 13) top rolls of three-high stands.

The roll setting crew prepare the equipment necessary for changing, repairing and preparing new guides, repairing and replacing the guards, hangers and bolts, checking the quality of repair of the guides and their dimensions (length, width and height). After checking, the guides are assembled complete, set up in pyramids in the gangways and marked with chalk.

In addition to the rolls and equipment, the necessary tools are got ready for roll changing.

The main preparatory operation to roll changing is the complete assembly and adjustment of the three-high stands I and II on the platform for spare stands, also the assembly of the top roll and cover plate (cap) of two-high stand III.

Six men are employed in assembling the stands: a foreman roll setter, two roll setters, a fitter and two electric crane operators.

Their duties are distributed as follows:

The foreman roll setter supervises the work of assembling and setting up the roll stands in accordance with the roll designs, selecting the guides and mounting the hangers and guards.

The roll setters locate, assemble, adjust and fix the new rolls and equipment.

The fitter is responsible for the preparations for assembling the rolls and chocks; he removes dirt from them, applies a thick layer of grease to the roll necks, bearings and side faces of the chocks, replaces unsuitable textolite bearings and helps in assembling the top rolls and housing cover plates of stands I, II and III.

Assembling the stands on the platform. Assembling of the separate stand units is carried out in three places by the two electric cranes. The following is the sequence of operations:

1. The necks of the bottom and middle rolls of stands I and II are greased.

2. The middle rolls of stands I and II and the chocks are assembled. At the same time, the chocks of the bottom rolls are adjusted.

3. The bottom rolls are located in stands I and II, the leading roll setter ensuring that the rolls are put in the correct positions and are properly marked on the end collars facing the main motor (bottom roll one mark, middle roll two marks and top roll three marks).

When using a unit having a second middle roll, when the top roll is mounted at the bottom and the bottom roll at the top, the rolls should be marked as follows: bottom roll three, middle roll four and top roll one.

4. The bottom hangers (brackets) on both sides of the stand are placed in position.

5. The middle rolls in stands I and II are placed in position, as well as the entry guides between the top and middle rolls, and the bottom hangers are secured.

6. The middle and top hangers and guides are placed in position on stands I and II.

Simultaneous assembly of two stands saves time in the individual operations, since no time is lost in changing the slinging gear and in unnecessary crane movements.

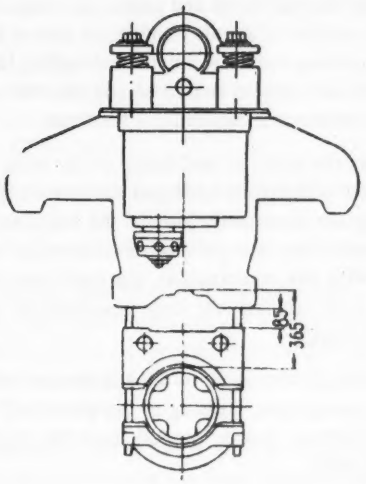


Fig. 2. Construction of cover plate of two-high stand as assembled.

7. During assembly of the rolls and hangers and the mounting of the guides on stands I and II by crane No. 2, the top rolls and cover plate of stands I, II and III are assembled on additional platforms by crane No. 1.

8. The cover plates with the top rolls assembled are carried by crane No. 1 to the platform for spare stands and are mounted on their stands.

9. The top suspended guides are mounted.

10. The housing cover plates are secured by wedges designed to prevent the cover plates from being lifted with the top rolls during passage of the rolled piece.

11. The middle rolls are secured provisionally by wedges. The wedges are finally tightened on the passage of the piece between the middle and top rolls.

12. The gaps along the passes and end collars are fixed in accordance with the roll designs. Allowance is made for the give of the rolls during the passage of the piece, this being assumed to be 2.5 – 3.0 mm for stand I, 1.5 – 2.0 mm for stand II and 1.0 mm for stand III.

When the rolls are mounted horizontally in the stands, the clearance at the tapers should be 1.5 mm by gage (for tapers which operate during the passage of the piece through the pass); in other cases, mounting of the rolls should ensure symmetry of shaped sections; consequently, the rolls are mounted in accordance with templets.

After setting (adjustment) of the rolls in the vertical and horizontal directions, the accessories (hangers, guards and guides) are mounted and secured.

The stands are then considered to be provisionally adjusted.

To speed up the adjustment of the two-high finishing stand, the housing cover plate should be assembled so that the distance between the roll neck and the flat for the nut of the pressure-adjusting screw, for new rolls, is 365 mm or 85 mm from the base of the cover plate ("cap") to the top of the chock (Fig. 2). For rolls of smaller diameter, this distance is increased to the total difference between the roll diameters (since the bottom roll has no vertical adjustment).

Roll changing on the mill. To begin with, the wedges fixing the housing cover plate (cap) are knocked out on the finishing stand. Crane No. 2 removes the cover plate from the stand and carries it to the platform. On its return trip, the crane picks up the previously prepared cover plate with the new top roll assembled. In the meantime, crane No. 1 removes the hangers (brackets), takes out the bottom roll and then puts in a new bottom roll and hangers.

Crane No. 2 places the cover plate with top roll on the housings of the finishing stand. During roll changing of the finishing stand, preparations are made for changing the rolls of stands I and II; the lifting

Foreman P. D. Krishtofovich making a test measurement.

tables are moved away, the wedges fixing the stands to the bedplates are knocked out, the coupling boxes are taken off, the water hoses and spray manifolds for cooling the roller necks and passes are disconnected. Crane No. 1 then removes stands I and II and sets them down on the spare stand platform; on the return trip, this crane picks up the new stands prepared for rolling a fresh section.

After this, the water hoses and spray manifolds for cooling the roll necks and passes are connected up, the fixing wedges of stands I and II are driven home, the lifting tables are moved up, the coupling boxes are put on, the water is turned on and the 6200 and 1500 h.p. motors are started for a trial run.

After the trial run and fixing of the rolls, the foreman and senior mill hand test the water supply for cooling the necks and passes of the rolls, and then give the order for testing the remainder of the mechanisms. With this organization, the total time for roll changing is 58 minutes (the time according to plan is 1 hour 5 min.).

Adjusting the mill. Foreman P. D. Krishtofovich pays particular attention to the adjustment of the mill equipment, since incorrect entry or delivery of the piece results in stoppages, chilling of the piece and in some cases damage. The mounting and fixing of the equipment (hangers, guards, guides) should be such as to ensure correct entry of the piece and straight delivery from the rolls.

The entry guards should be mounted so that when the piece is fed to the pass, it does not jam in the guards, otherwise the end of the piece may be chilled with water and the rolls may be fractured. Usually, the entry guards are mounted at a distance apart equal to the width of the preceding pass.

The guides should be of a definite length. Short guides can be pressed down if the pressure exerted by the piece is high; if they are too long, they are easily deflected.

The leading crew of the rolling mill, under the supervision of Foreman P. D. Krishtofovich, by employing the above sequence in carrying out the job of roll changing, have reduced the total time for roll changing by 7 min., due to the use of internal productivity reserves, have ensured early completion of the year's plan and have rolled 7161 tons of metal over and above the plan.

The adoption and introduction of advanced methods in roll changing by all crews enables up to 30 hours of working time to be saved per annum and output to be increased by 2 or 3%. In this way, additional thousands of tons of steel can be rolled annually.

NEW BOOKS

IRON PRODUCERS CRITICIZE SCIENTIST'S BOOK

The blast furnace plant section of the scientific and technical department of the Magnitogorsk Metal Combine has criticized the book by M. Ya. Ostroukhov "Forcing blast furnace operation" (Metallurgy Press, 1956). Blast-furnace foremen, plant workmen and the Mining and Metallurgical Institute contributed to the criticism of the book.

After appreciating the work of the author in collecting together a large quantity of factual material and the results of investigations carried out on blast furnaces at different works, the contributors of the criticism indicated a number of vital defects, considerably reducing the value of the book.

The the opinion of Comrade Ryabtsev, the author has adopted criteria for forced blast-furnace operation which are not characteristic of the blast furnace: quantity of coke burnt and air blown in. These criteria are more characteristic of the gas-producer process. Increasing the quantity of air blown in disturbs the working of the furnace, since with the high density of the modern burden, it is very difficult to blow air through the column of material.

Foreman Morev had much to criticze in the second chapter of the book. In considering the mechanical processes taking place in the blast furnace, the author has made a large number of unnecessary and trivial calculations. In addition, certain hypotheses expressed by the author in this chapter contradict practice at the Magnitogorsk blast furnaces. For example, the author asserts that steam is introduced into the blast in order to compensate for increased heating of the blast. Magnitogorsk blast furnacemen consider that steam is a means of regulating the thermal conditions of the furnace. Thus, reduction in the amount of moisture in the blast, in the case of basic slags, assists in forcing the operation of the furnace. The data quoted by the author in connection with the oxidizing zone are old. Certain questions of considerable interest to blast furnacemen require elucidation, such as the thermal condition of the hearth, blocking of the hearth, methods of ascertaining the commencement of channel formation, etc. Slag conditions, desulfurizing capacity of the slag and the function of manganese, should have been discussed in more detail.

In the fourth chapter "Blast-furnace profile", as Comrade Mishin remarked, the author incorrectly states that a wide bosh is necessary for economic working of the furnace. In the Magnitogorsk furnaces, wide boshes as a rule readily give rise to peripheral gas flow. The furnace profile ought to correspond to the melting conditions and particularly to the degree of preparation of the raw materials. Other defects were also noted. For instance, in the fourth chapter, the statement regarding the gap between the wall of the furnace throat and the large bell is obscure. Cooling of the throat with water as proposed by the author cannot be recommended: satisfactory operation of the throat apparatus must necessarily be ensured by maintaining an even temperature of the throat and by uniform operation of the furnace.

In criticizing the fifth chapter, Comrade Kryukov remarked that the data regarding the intensity of combustion and the coke consumption are contradictory. The author does not define the optimum time of stay of the materials in the furnace. Comrade Kryukov pointed out that the preparation of the burden materials is of very considerable importance for the reduction process.

In the opinion of Comrade Ryabtsev, the operation of blast furnaces with increased blast pressure and wet blast (sixth chapter) should not be related to special forms of melting. The author's recommendation to employ acid slags with a basicity of up to 0,5 cannot be accepted, since with such basicity, it is impossible to remove sulfur to the necessary extent. At the same time it was noted that the author recommends increasing the capacity of slag ladles and iron ladles.

After remarking on the considerable amount of work done by the author in studying the data of various investigations and experiments, Comrade Stefanovich expressed regret that the abundance of examples and references prevented one from perceiving the fundamental ideas developed by the author. Often the author fails to give adequately complete data. For example, referring to the intensive operation of blast furnace No. 5 at the Magnitogorsk Metallurgical Combine in 1945, the author omits to cite data regarding the condition of the materials and their content of lumps and fines. Referring to the data of G. G. Oreshkin on the utility of widening the bosh, he fails to produce any concrete practical data.

The author does not express any view at all on many questions; some questions are incorrectly treated (for example that concerning the increase in the zone of slag formation with increase in the blast temperature).

Unfortunately, the book published in 1956 does not contain any information on recent blast furnace operation and results of investigations (later than 1952-1953).

SCHOOLS OF ADVANCED EXPERIENCE

ADVANCED EXPERIENCE IN THE ORGANIZATION OF BLAST-FURNACE REPAIRS

Engineer D. I. Felman

Senior Engineer of the Department of the Chief Engineer of the

Ministry for Ferrous Metallurgy of USSR

An inter-works school for the study, generalization and introduction of advanced experience in blast-furnace repair and the care and maintenance of blast-furnace equipment was set up at the blast-furnace plants of the following metallurgical works: Dzerzhinsk works, "Zaporozhstal," Kirov Makeev works, "Azovstal", Novo-Tagil works and the Magnitogorsk Metallurgical Combine. The achievements and shortcomings at each of these works were noted and recommendations made for improving the operation and repair of the blast-furnace plant equipment.

A description follows of the advanced experience in the organization of blast-furnace repairs according to the information collected by the school.

The change-over of blast furnaces to operation with increased gas pressure below the throat, increase in the blast temperature to 800 – 900°C, introduction of hot sinter in the charge has meant a sharp increase in the intensity of the blast-furnace process. This has been accompanied by a considerable reduction in the durability of the equipment, especially the charging apparatus, hot blast valves, tuyeres, throat armoring, etc.

Depending upon the life of the individual parts and units, the frequency of stoppage of the blast furnaces for repair varies within the following limits: a) for replacing the small bell – from 6 months to one year: b) for replacing the charging apparatus – from 8 months to $2\frac{1}{2}$ years; c) for capital repair with replacement of the shaft lining, cooling plates and charging apparatus – from 2 to 5 years; d) for capital repair with replacement of the well and hearth – from 5 to 10 years.

At the leading works, Magnitogorsk Metallurgical Combine, "Azovstal," "Zaporozhstal" and Novo-Tagil works, a considerable amount of work has been done on improving the construction of the equipment and selecting wear-resistant materials; by heat-treatment and hard-alloy facing of parts, the service life of the

equipment has been increased in the new conditions.

On the basis of the experience of the advanced works, the school recommended the adoption of the following frequency for blast-furnace repairs: a) capital repair with replacement of well and hearth – 8 to 10 years; b) capital repair with complete replacement of shaft lining, cooling plates and charging apparatus – 4 to 5 years; c) capital repair with replacement of the charging apparatus alone – 2 to 2.5 years.

At the present time, blast-furnace repairs are carried out in accordance with previously drawn up programs of working organization, devised for the application of industrial methods and advanced technology, the assembling of steel constructions, castings and equipment by large units and the multistage execution of dismantling and assembling work.

It has been found that for carrying out rapid repairs special importance attaches to early and qualitative preparation of steel constructions, castings, equipment and other material, a definite organization program of the work and the proper preparation of the blast furnace for blowing out.

For this purpose, it is recommended that in the work organization programs, consideration should be paid to the obligatory and controlled assembly of large constructional and equipment units, and in particular items of the charging apparatus in the following units: large bell and hopper, hopper and gas seal, gas seal with charge distributor, bell rods and links. Controlledassembly should ensure that the parts of the charging apparatus are perfectly coaxial. Balancing and inspection of the large and small bells should be done in the machine shop on a layout block and a special column, as shown by experience at the "Zaporozhstal" works. To enable the charging apparatus to be replaced in large units, the load-bearing capacity of the assembling girders should be increased in time for the repairs: for furnaces with a capacity of 1386 cubic meters – up to 80 tons, a capacity of 1033 cubic meters – up to 60 tons and a capacity of 700 cubic meters – up to 40 tons.

It is recommended that there should be a centralized supply of oxygen, compressed air and mortar. Stocks of refractories should be sited so that the refractories can be supplied to the working places without the need for reloading and offloading. Considerable importance is attached to rationally directing the supply traffic, so that it is possible to carry out simultaneously assembly and refractory work at several stages.

The proper preparation of the furnace for blowing out presupposes the removal of slag and iron before shutting down the furnace for repair, running off the salamander in molten form and a copious supply of water to the furnace during the first day of repairs. If these steps are taken, the time taken for capital repairs can be reduced by 10 to 15 days.

It is recommended that the salamander should be run off molten through special tap holes: for furnaces of large capacity, two salamander tap holes should be provided, for furnaces of a capacity less than 700 – 900 m^3, there should be one tap hole. Through the upper salamander tap hole, the iron should be run off into iron ladles, and through the lower tap hole, it should be run into specially prepared, lined boxes mounted on trough bogies (Fig. 1).

Water should be run into the furnace for 18 to 24 hours. Copious application of water, in consequence of the action of water and steam, assists rapid destruction and breakdown of crusts and the refractory lining of shaft and hearth.

The construction and the arrangement of blast-furnaces, equipment and axiliary structures at the works of the Magnitogorsk Metallurgical Combine differ considerably. As a consequence of this, different methods and procedures have been developed at the works and in the "Uraldomnaremont" and "Yuzhdomnaremont" trusts for carrying out repairs.

Investigation showed that the best methods and procedures for carrying out repairs employed at a number of works and in the constructional departments of the "Uraldomnaremont" and "Yuzhdomnaremont" trusts can be introduced into all the works of the Ministry.

The following is a description of the fundamental methods and procedures for the repair of furnaces as recommended by the school.

The "Yuzhdomnaremont" trust has developed a new technique for the removal of metal salamander. The essential feature of this technique is as follows. In the first days of the repair, four large, diametrically

situated openings or windows are cut out in the shell and cooling plates of the hearth. The contour of the salamander, the temperature of which attains 800 – 900°C, is ascertained, the lining round the salamander is then removed and horizontal shot-holes of 100 to 120 mm are burnt in the body of the salamander by means of oxygen to a depth of 1.5 to 3m. Water is continuously supplied to the holes for cooling the salamander. Vertical holes are then burnt through by means of oxygen. Ammonite, detonator caps and detonator fuse are required for shotfiring.

After shotfiring, the lumps of salamander are removed by means of block and tackle and winches. Figure 2 shows the various stages in carrying out this work.

The dismantling and erection of the charging apparatus is carried out in large units. The large bell is fixed to the hopper by strips and all bolted joints are cut. At the same time, the bleeder valves are dismantled and then the receiving hopper. The distributor and gas shut-off device are likewise dismantled in large units with the severed portions of the large and small bell rods and the unit comprising the hopper and large bell.

Erection of the charging apparatus is carried out in two stages. A unit consisting of the large bell and hopper is mounted on the supporting ring which has been left in position; then the second unit, consisting of the gas seal, distributor and rods of the bells and small bell, is hoisted into position. Such a method of dismantling and erecting the charging apparatus enables the latter to be replaced in short periods of two days.

In replacing the charging apparatus and armoring segments, it is recommended that use should be made of the experience of the Dnepropetrovsk constructional department of the "Yuzhdomnaremont" trust. At the same time as the charging apparatus is dismantled, openings are cut in the furnace shell on the charging hoist side below the level of the segments and the latter are removed through these openings. Batches of segments are lowered through the top ring on to a platform suspended inside the furnace and are then installed. The remaining segments are delivered from the charging hoist side through openings cut in the shell. Such a method of carrying out the work in two simultaneous stages ensures a reduction in the time required for repairs. Figure 4 shows diagrammatically the organization of this work.

When carrying out capital repairs with complete replacement of the shaft lining, armoring segments, cooling plates, boshes and charging apparatus, it is recommended that the work should be carried out in multiple stages.

During the period of repair of the charging apparatus, four steel strips are lowered through openings previously cut in the crown of the furnace and three steel platforms are suspended from these strips. The top platform is mounted on a level with the segments, the middle platform on a level with the "false" mantle and the bottom platform on a level with the main mantle. These platforms serve for carrying out the work of dismantling and restoring the lining.

The work of restoring the lining inside the furnace is carried out in four stages: the first stage – boshes, second stage – from the main mantle to the "false" mantle, third – from the "false" mantle to the segments and the fourth – zone of the segments and crown. Installation of the armoring segments, charging apparatus and cooling plates proceeds in parallel with the lining of the shaft. The total length of repairs of a large blast furnace from blowing out to blowing in is 12 to 13 days if the work is carried out in stages. Figure 4 shows diagrammatically the organization of this work.

In the case of capital repair with replacement of the bottom and hearth, running off the molten salamander, breaking up and removal of solid salamander, dismantling and erecting of the charging apparatus are carried out in accordance with the technique described above.

The dismantling work inside the furnace is carried out in three stages: dismantling the hearth, dismantling the refractory lining of the shaft, dismantling segments and removal of the brickwork in them.

So that the work of dismantling can be commenced as soon as possible, a strong sheet-metal partition is mounted on a level with the tuyere zone and is provided with one or two chutes. By this means, it is possible to dismantle the refractory lining of the shaft at the same time as the work in the hearth. As the bricks fall on the partition, they descend by gravity to a conveyor or directly into railroad trucks. The capital repairs for a furnace of large capacity take 30-33 days. Figure 5 shows diagrammatically the organization of this work.

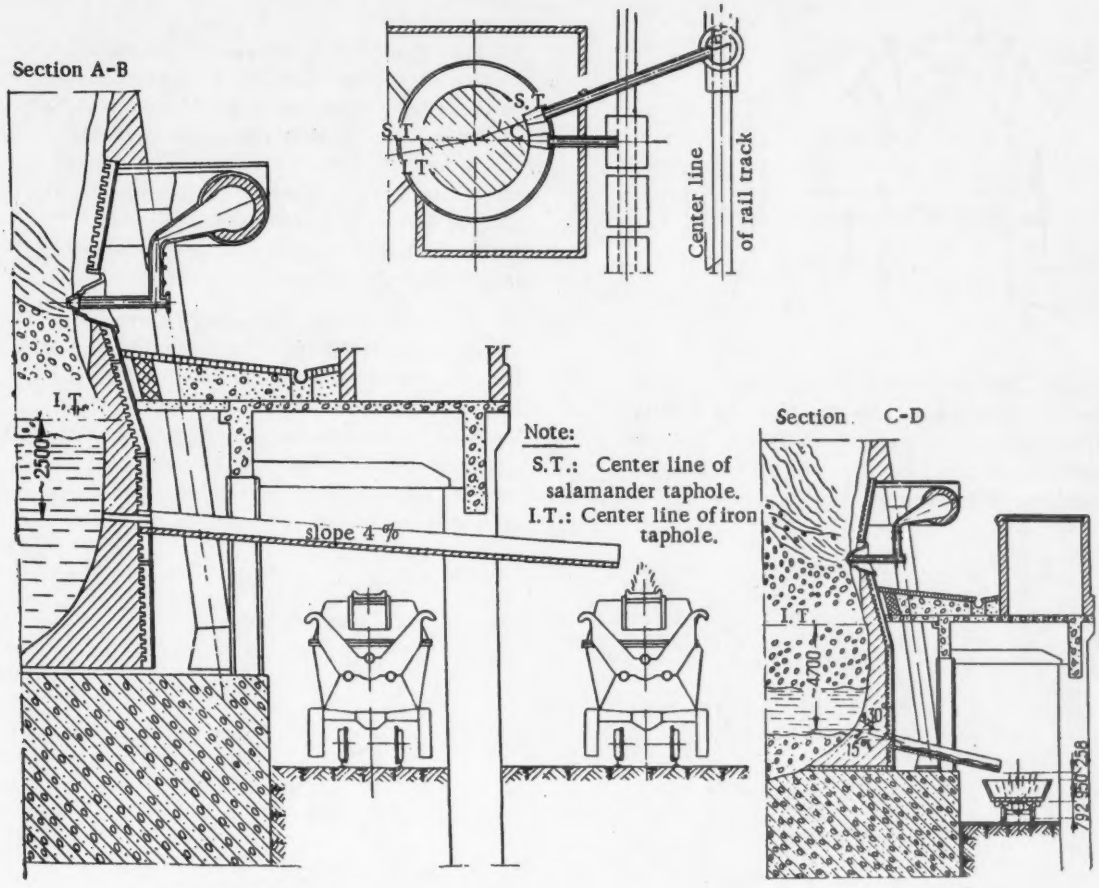


Fig. 1. Running off the molten salamander.

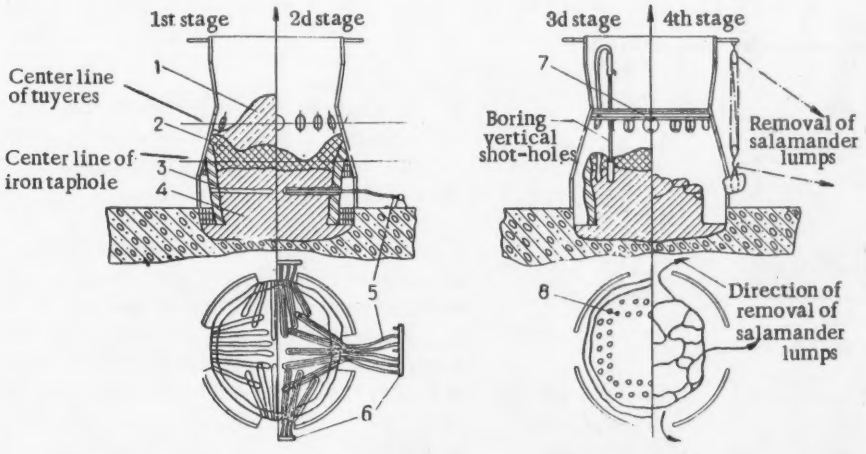


Fig. 2. Breaking up and removal of solid salamander. 1) charge; 2) conglomerate; 3) horizontal shot-holes; 4) solid salamander; 5) spray device for water-cooling the shot-holes; 6) manifolds; 7) sheeting; 8) arrangement of vertical shot-holes.

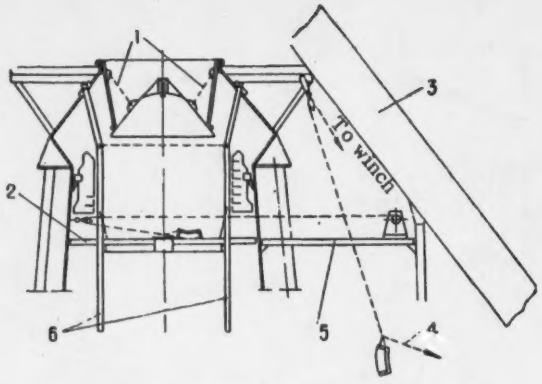


Fig. 3. Diagram showing the organization of the work of replacing the charging apparatus simultaneously with the armoring segments: 1) ties; 2) suspended platform; 3) charging hoist; 4) guy; 5) temporary platform; 6) steel strips.

In reconstructing blast furnaces, it is most expedient to replace the old furnace by a new one by a process of removal. The new furnace (all the steel construction, castings, equipment and cooling water connections) is erected on one side before blowing down the furnace in service and is moved on to the previous foundation by means of winches with a block and tackle system along a special stand or on a platform of sleepers (Fig. 6). The old furnace is dismantled in large units. Such a method of erection enables a furnace of large capacity to be reconstructed in 45 – 50 days.

With the object of reducing the time spent on repairs and preserving the construction of blast furnaces, the school recommended that the state Institute for the Planning of Metallurgical Plants the state Institute for the Planning of Steel Constructions and the metallurgical works should consider the following constructional details and devices in their projects.

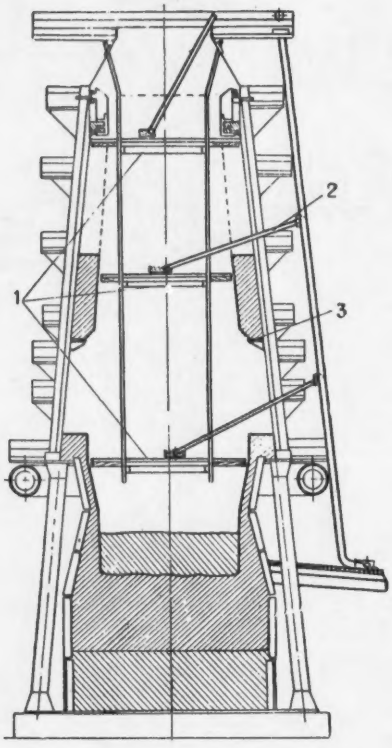


Fig. 4. Diagram showing the organization of the work in the capital repair of a furnace with replacement of the shaft lining, armoring segments and charging apparatus. 1) suspended platforms; 2) belt conveyor; 3) "false" mantle.

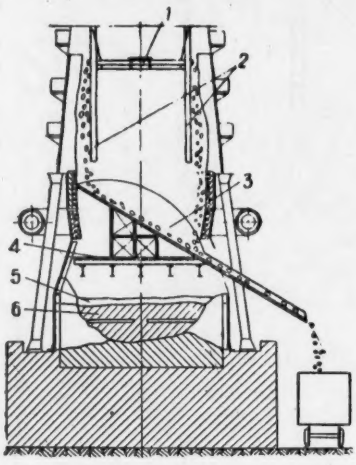


Fig. 5. Chute for removing bricks falling from furnace shaft. 1) suspended platform; 2) steel strips; 3) chute; 4) sheet partition; 5) conglomerate; 6) solid salamander.

Large bell, hopper and gas seal of charge distributor should have three assembly lugs each and the strengthening bell and small bell should have two each. The lugs should be designed for lifting the parts in large units. Permanent assembly lugs should be on one of the safety rings of the rod of the small bell. The diameter of the orifice in the lower flange of the gas seal should be increased by 10-15 mm in comparison with the orifice in the supporting ring of the throat. The receiving hopper should have a detachable insert on the charging hoist side. In the construction of the

seal of the charging apparatus, a "floating" flange should be provided, permitting rapid and accurate self-centering of the small bell and the rotating hopper of the charge distributor.

The shell of the hearth should be provided with two tap-holes with built-in coolers for running off the molten salamander: one at a level of 2 – 2.5 m below the center line of the iron tap-hole for running the salamander into ladles, and the other on a level with the bottom course of the furnace well for running the salamander into boxes mounted on steel distributing bogies.

At three or four levels in the shaft shell, openings (manholes) 800 x 600 mm with bolted-on flanged covers should be provided to eliminate the need for cutting through the shell during repairs for supplying bricks and removal of accretions. The projects should also provide for detachable flanged plates with coolers: two plates 1.5 x 2 m in the boshes for removal of large lumps of accreted material and dirt during capital repairs, two plates in the tuyere zone for the same purpose, four plates 1500 x 1500 mm in the hearth for removal of salamander and dirt resulting from the stripping of the masonry of the hearth and bottom. At each level, one of the openings should be on the center line of the iron tap-hole.

In the working area opposite the detachable plates of the tuyere zone, openings should be provided, adapted to be closed by reinforced concrete slabs 1500 x 1500 mm for erecting chutes when removing dirt from the furnace.

In the shell of the hot blast stoves, special manholes with detachable covers should be provided at a distance of 5 – 7 m apart for removing fallen masonry when stripping the lining or for supplying bricks for repairs.

The walls of spaces below the bunkers should have openings left specially in them, these being uncovered during repairs for delivering bulky parts and assemblies to the skip pit of the bunkers.

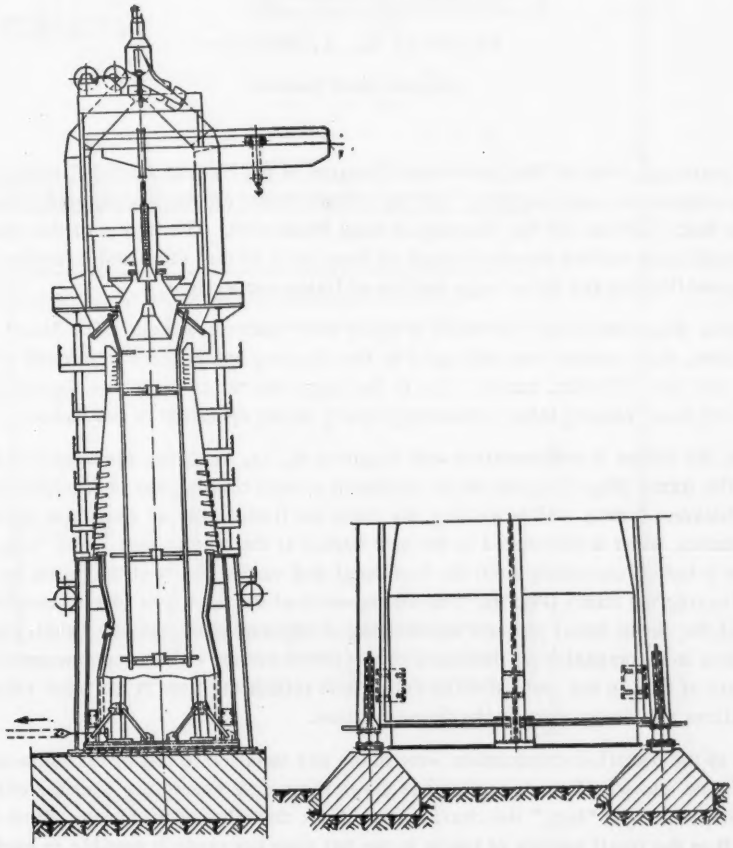


Fig. 6. Moving a blast furnace into position.

The cooling plates should be cast with minus tolerances ensuring the formation of vertical and horizontal gaps of 40 mm; each row of cooling plates should include a locking cooling plate of special construction to facilitate installation and dismantling (the locking cooling plate is marked with the sign Z).

The overhang of the erecting girder should be increased; this will enable parts to be hoisted without the need for guys.

It appears expedient to erect a mono-rail round the furnace below the bustle pipe for dismantling and erecting the tuyere equipment. The mono-rail should exit on the pig bed within the service zone of the overhead crane. For convenience in installing the shaft cooling plates, a mono-rail with crane should be erected round the furnace below the top platform.

The school recommended the design and erection outside the furnace of two vertical pipes of a diameter of 600 mm with branches for the removal of dirt during lining work (one on the charging hoist side and the other on the erecting girder side). These pipes could be used also when cleaning the shaft platforms during operation of the furnace.

NEW TECHNIQUES AND RATIONALIZATION

NEW TYPE OF DOOR FRAME FOR OPEN-HEARTH FURNACES

Engineer G. A. Sokolov

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A few years ago, some of the open-hearth furnaces of the "Krasny Oktyabr" works were converted for operation with evaporative water-cooling. All the constructional elements were made in accordance with the drawings of the State Institute for the Planning of Steel Production. The frames of the charging doors (Fig. 1) represented complicated welded structures made of sheet steel 12 mm thick and comprised elements necessitating bending and flanging and had a large number of fixing connections.

In service, this construction was found to suffer from many drawbacks. The "feet" of the frame had a rectangular section, their corners were damaged by the charging boxes, there were cases of welded seams being opened up and even of broken frames. Due to the large number of welds, the frames were often burnt through in the hot zone, causing labor-consuming repairs during operation of the furnace.

In 1954, the author in collaboration with Engineer M. Ya. Kapishko, proposed a new design of charging door frames. The frame (Fig. 2) represents an H-shaped welded construction of seamless tubes, diameter 219 mm and wall thickness 8 mm; before welding, the tubes are flattened to an oval cross-section 170 x 250 mm. As in the old frames, water is introduced in the new frames at the bottom part of the "legs," but the water-steam emulsion is led off separately from the horizontal and vertical parts of the frame by tubes connected together after leaving the frame (Fig. 3). The arrangement of the heat flow (due to suitable selection of the cross-sections of the outlet tubes) prevents overheating of any part of the frame. Greater heating of one of the frame sections is accompanied by increased convectional motion of the steam-water emulsion in it. Increase in the rate of flow in the corresponding outlet tube retards the flow in the other tubes and this automatically equalizes the temperature in the frame sections.

Frames of the described construction were made and installed in one of the furnaces of the steel plant in April, 1955. One and a half year's service has shown their great superiority over the old frames. Due to the oval cross-section of the "legs," the charging box slides through without tearing off or damaging the frame. This fact as well as the small number of seams in the hot zone has made it possible to avoid repairs to the

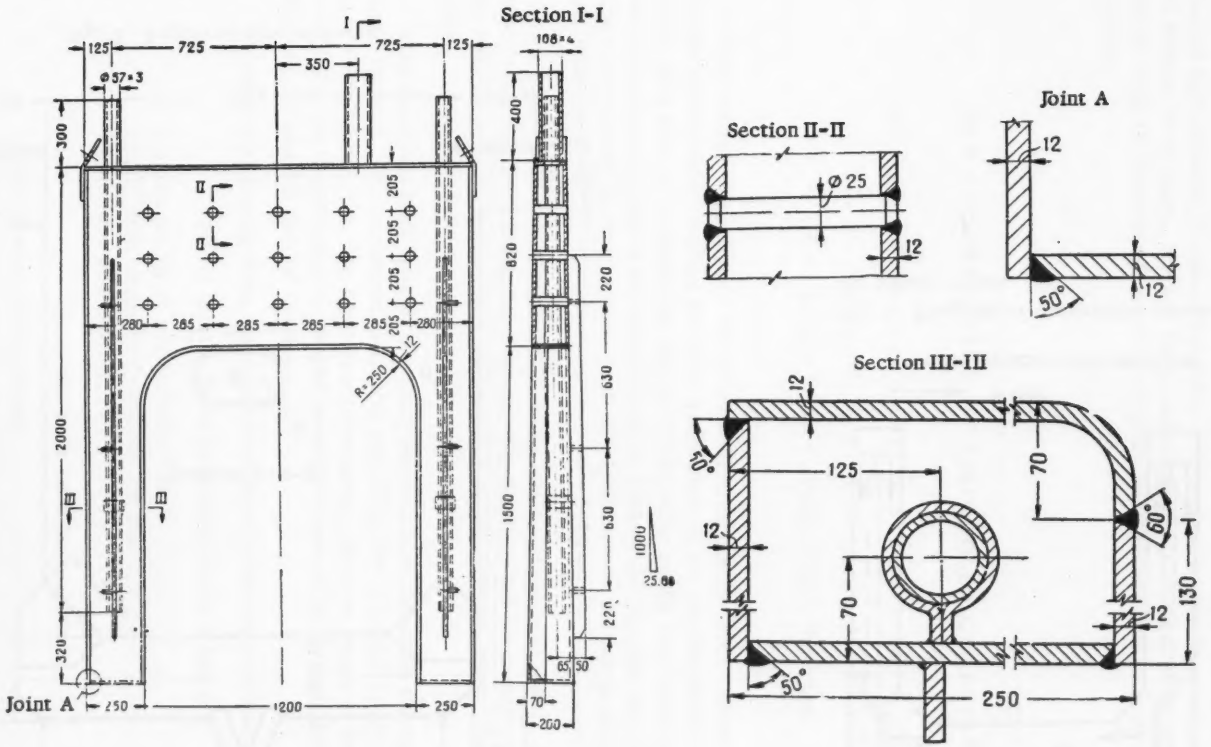


Fig. 1. Frame according to the design of the State Institute for Planning Steel Production.

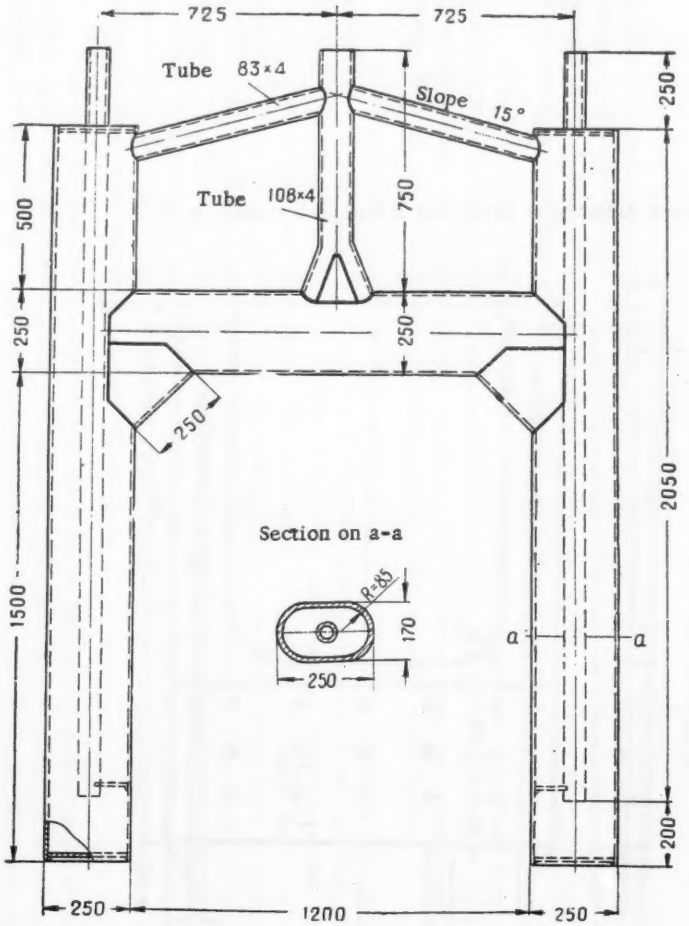


Fig. 2. Improved type of frame.

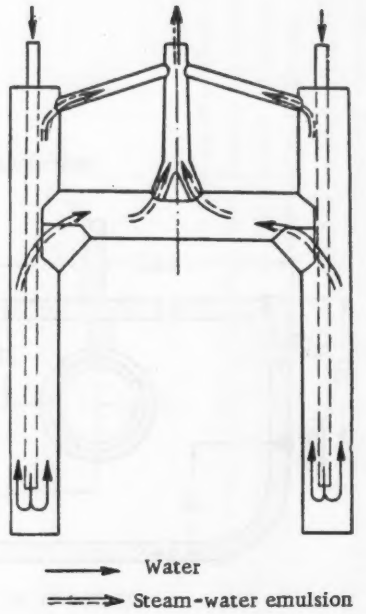


Fig. 3. Diagram of convection currents in the tubular frame.

frames during the furnace operation. According to preliminary data, the service life of the tubular frames exceeds that of frames of the earlier design by at least 7 to 8 months, i.e. by 80 – 90%.

The superiority of the new frames is more pronounced particularly with increase in the pressure in the evaporative cooling system from 2 to 4.5 atm. gage pressure. The form of the frame elements, the absence of a water reservoir in the upper part and the small number of welds make these tubular frames mechanically strong and capable of withstanding higher pressures.

The low weight of the tubular frames (301 kg against 595 kg for the old construction), simple manufacture and high reliability in service has led to their fairly rapid introduction in the majority of the open-hearth furnaces of the plant. The new frames are almost twice as cheap as the old ones. It is considered that equipment of all the open-hearth furnaces with these frames would give an annual saving of about 70 thousand roubles.

NEW CONSTRUCTION OF AIR-COOLED CHARGING DOOR

Candidate Technical Sciences S. M. Andonyev, Engineers

O. V. Filipyev, V. F. Volkov, B. M. Sirenko

State Institute for the Planning of Steel Production

Water-cooled charging doors for open-hearth furnaces suffer from a number of drawbacks: (1) Industrial water is used for cooling the doors, and at many of our metallurgical works, especially those in the Don Basin, such water is in insufficient supply; (2) the water carries away heat, resulting in unproductive fuel consumption and a reduction in the efficiency of the furnace; (3) the doors are often burnt through, since the cooling water is heated to temperatures above 40°C (with wear of the lining) and there is a considerable scale deposit inside the doors.

Air-cooled doors are undergoing trial at the Stalin Metallurgical Works. Such a door consists of a metal screening part and a lining slab (Figure). The slab is a steel box containing a dense reinforcement of steel rods of a periodic cross-section diameter 12 mm welded to the place. The triple screen is made of iron sheets.

When the slab is burnt through, it is easily removed from the screen and replaced by a new one when the door is repaired. The fundamental screening part of the door is subject to only slight thermal wear and can have a long service life. The lined plate (comprising 15% of the weight of the door in metal) is replaced as a whole after one month's service, despite incomplete wear of the lining (otherwise the screening part will be subject to considerable thermal wear).

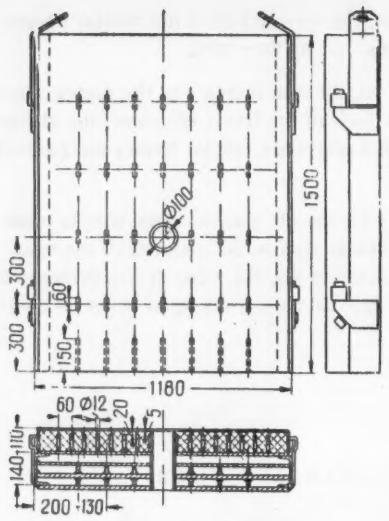
For lining the charging doors, mixes of two different compositions(percent)were used:

Mix of the All-Union Refractories Research Institute

Chrome ore (grain size less than 7 mm)	70
Highly burnt magnesite (grain size less than 0.88 mm)	30
10% sulfuric acid solution	10

Mix of the Stalin Metallurgical Works

Chrome powder (finely ground)	50
Magnesite powder (grain size less than 3 mm)	25
Chasov-Yar clay (finely ground)	25
Aqueous solution of alkali sulfide or water glass	6-7.5



Construction of air-cooled door.

Cost of different types of door in roubles (for one door)

Cost items	Door	
	Water-cooled	Air-cooled
Metal part of door	3060	1920
Refractory lining	2150	1000
Cooling water	3150	0
Total	8360	2920

During the trials, it was found that the mean service life of doors lined with the mix of the All-Union Refractories Research Institute was about 2 months, with the lining worn to 30 – 40 mm. The service life of doors lined with the mix of the Stalin Metallurgical Works was 24 hours on the average, the lining being worn to 10 – 20 mm. With such wear of the lining, the metal part of the doors is considerably distorted; during repairs, the door body has to be straightened and 30 – 40% of the metal replaced.

Preparation of the lining from the mix of the All-Union Refractories Research Institute, fine grinding of the highly burnt magnesite (periclase cement) is essential and this can only be done in vibratory mills (for instance, of the VM-200 type). The mix of the Stalin Metallurgical Works can be made without difficulty in any refractory grinding and mixing shop.

The lining made from the mix of the Stalin Metallurgical Works is less durable than that from the mix of the All-Union Refractories Research Institute.

In the use of the doors at the Stalin Metallurgical Works, it was found that the average temperature of the outer screen was 50°C. After considerable wear of the lining (towards the end of the month after installation), the temperature of the outer screen rose to 80°C.

The heat loss through the air-cooled door in the first few days after installation is 600 cal/hour; towards the end of the month when the lining has become considerably worn, the heat loss is increased to 15000 cal/hour.

The figures given in the table show the economic advantage of using air-cooled doors.

In conclusion, it should be mentioned that the use of air-cooled doors with rammed lining and sheet-metal screening does not interfere with the operation of the furnace, excludes the possibility of water getting on to the roof and front of the furnace and reduces the consumption of water for cooling the furnace.*

* Publisher's Note: The air-cooled door which has been tried at the Stalin Metallurgical Works still has some substantial defects: (1) In addition to the wear of the lining, the metal part to which the rods are welded also wears and its repair merely by welding on rods does not appear to be possible. The wear of the doors is particularly high at the position of the inspection hole and in the lower part; (2) the temperature of the outer screen of the uncooled door is considerably higher than that of the water-cooled door, especially in the summer time (reaches 150°C).

The construction of the air-cooled door requires further improvement and the development of a special lining outside the open-hearth steel shop.

METALLURGY ABROAD

AT THE CONFERENCE OF GERMAN ENGINEER-METALLURGISTS

Engineer A. K. Adabashyan

In October, 1956, in Leipzig, a conference was held by the Society of German Engineer-Metallurgists (G.D.R.), in which 700 persons took part, including representatives of the USSR, the Chinese People's Republic, the Korean National Democratic Republic, Bulgaria, Rumania, Hungary, Poland, Czechoslovakia, India, Denmark, Belgium, Sweden and West Germany. The Conference had four sections: Ore-mining, ferrous metallurgy, non-ferrous metallurgy, mechanization and automation. At the plenary and section meetings, 19 papers were read and there were three communications from representatives of the USSR delegation.

The paper by Professor Doctor L. Born "Principles in the use of radioactive substances in research work and technology" was devoted to the possibility of producing highly sensitive instruments for making observations on technological processes by means of radioactive isotopes. Special attention was paid in the paper to the problem of the safety of working with radioactive substances. L. Born and other authors consider that it is necessary to develop and approve universally compulsory, international rules for safety techniques in working with isotopes.

This is very important because countries which are beginning to use isotopes for the first time possess no experience and may cause harm not only to their own country but also to neighboring countries (for example, by the contamination of rivers passing through the territory of several countries, etc).

K. Leitner, in his paper, described an interesting experiment in the use of basic refractories for lining a 150-ton open-hearth furnace, constructed according to the plans of the Austro-American firm "Magnesit" at the Brandenburg Metallurgical Works (G.D.R.).

All the most important lining elements of the open-hearth furnace which uses the scrap process with cold charging, are made of magnesite-chromite bricks. The roof of the working space is lined with a ribbed magnesite-chromite brick of the mark Radex E (Fig. 1, 2). The simplicity and relative low weight of the suspended roof construction is of particular interest (Fig. 3). The sloping parts of the vertical gas and air flues and the roofs of the slag pockets and regenerators are also lined with magnesite-chromite bricks. For providing favorable aerodynamic conditions for the flow of the gas, air and products of combustion, the masonry of the inlet portions from the generators to the slag pockets and from the slag pockets to the vertical flues and also into the gas port are streamlined. This has been made possible by the suspended brickwork system, which as the speaker confirmed, has given good productive results: increased life and density of the brickwork, reduced consumption of refractories, and improved the distribution of the exiting combustion products between the gas and air flues.

In the case of need, with flat suspended roofs of the regenerator chambers, it is possible without considerable expenditure to join together two chambers on one side of the furnace and thus to increase considerably the heating surface of the checkers.

The top 17 courses of the air checkers are of magnesite-chromite bricks, 15 courses being on the Cowper system with a cell cross-section of 100 x 120 mm. The lower courses of the air checkers are made of fire-bricks, containing 40% Al_2O_3. The gas checkers are made of a specially shaped Dinas brick (Figs. 4, 5), which is a square tile 60 x 240 x 240 mm. In the middle of the tile is an opening 100 x 100 mm. According to statements of the workers, these checkers have resulted in improved heating of the gas.

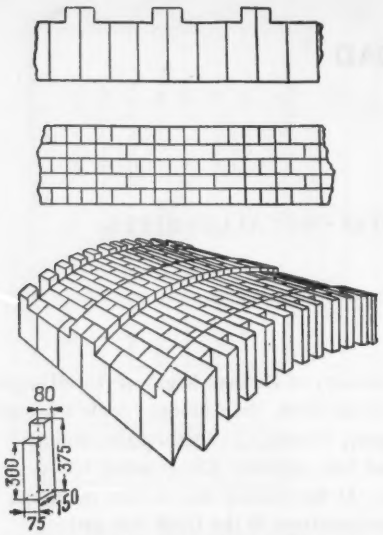


Fig. 1. Brickwork of roof of working space of open-hearth furnace.

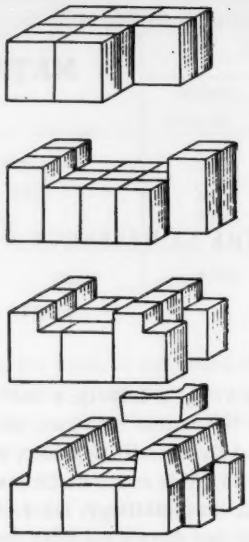


Fig. 2. Radex E brick.

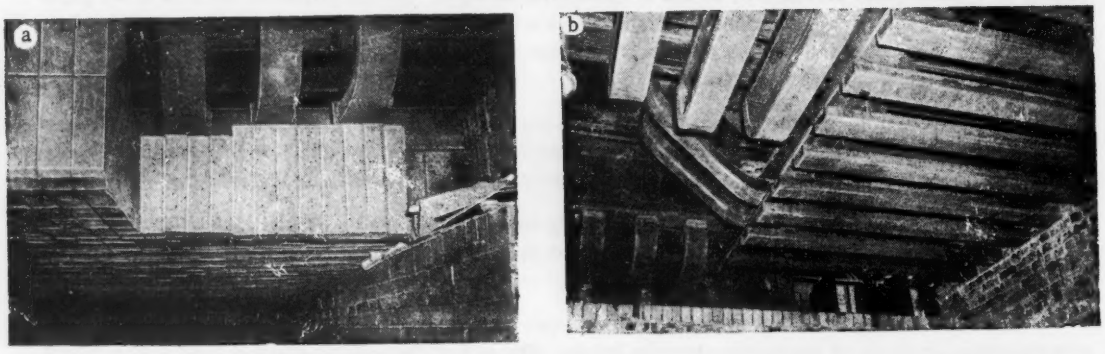


Fig. 3. Suspended roof of open-hearth furnace a) brickwork of roof; b) metal construction of roof.

From the time of starting up, 112 melts have been produced in the all-basic furnace. The state of the gas and air checkers and also the masonry of the working space is good. The average hourly production of the furnace was 12.75 t/hour in October. Compared with the average production of the steel plant, melting in this furnace has been reduced by one hour (11 hours instead of 12).

In the paper by R. Baake and Engineer H. Stollberg "Thermotechnical investigation of open-hearth furnaces" copious data were produced on the form and movement of the flame in industrial open-hearth furnaces with Merz ports (Fig. 6) at the Brandenburg Metallurgical Works. During the reading of the paper, a colour cinefilm was shown, illustrating clearly the effect of certain modifications of the qualitative characteristics of the producer gas and of the fundamental dimensions of the furnace ports on the shape of the flame in the melting space.

On the basis of these investigations, the speaker showed that the siting of the air flues close to the gas port has a favorable effect on the aerodynamics of the flame and combustion processes of the gas. The velocity of the gas leaving the gas port, for Merz ports, should be preferably 45 – 50 m/sec, and that of the air leaving

the air ports 15 – 20 m/sec. Increasing the air velocity disorganizes the aerodynamics of the flame, the gas stream being deflected from the longitudinal axis of the furnace.

It should be noted that the method of studying the flames of reverberatory metallurgical furnaces by means of colored cinematography deserves considerable attention and should be adopted on an extensive scale in the USSR.

Engineer P. Perlik read a paper on automation in the metallurgical industry. He pointed out that in the automation of the blast furnace, three fundamental problems have to be solved: (1) enrichment and preparation of the burden; (2) uniform charging of the burden in the blast furnace; (3) thermotechnical control of the blast-furnace process.

The first two problems of automatic control have been solved only partly and the third has not been solved at all. The speaker referred to the positive experiment of the application of a complex scheme of controlling the thermal conditions by means of an electronic computer, as proposed by Professor Sorokin (USSR).

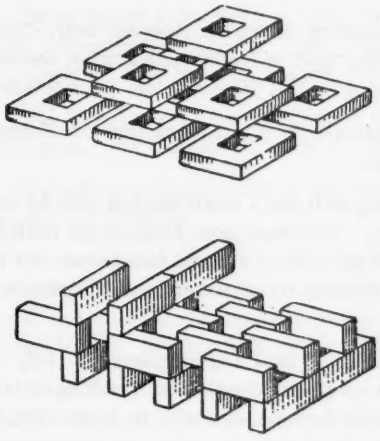


Fig. 4. Shaped Dinas brick.

The Soviet delegation was given the opportunity of visiting a number of works in G.D.R. A brief description of these works is given.

The Brandenburg Metallurgical Works has two principal metallurgical plants: the open-hearth steel plant and the rolling mill.

The open-hearth steel plant was started up in 1952. The plant has eleven 150-ton open-hearth furnaces operating on the scrap process with cold charging. The mean annual production of the plant is 850,000 tons of steel. All the furnaces are equipped with Merz ports and have Dinas roofs, with the exception of furnace No. 11 constructed entirely of basic refractories.

The open-hearth steel plant produces mainly plain carbon steels. Teeming in the casting bay is by bottom casting in ingots of 3 – 4 tons.

The average life of a roof is 112 melts. The monthly average of the length of a melting operation is 12 hours, which is made up of 3 – 4 hours charging, 5 – 6 hours melting, and finishing 3 hours. The plant is experiencing a serious shortage of steel scrap resulting in furnaces being closed down.

The mean thermal load is 14 – 15 million cal/hour. The furnaces are heated by hot producer gas, made from brown coal briquets. The calorific value of the gas is 1850 cal/nm^3.*

The gas-producer plant, situated at the side of the steel plant, is equipped with 44 gas producers having a diameter of 3.6 m and a height of 3.2 m., each with a mean capacity of 2200 kg/hour using brown coal briquets. The gas temperature at the producer outlet is 360 – 370°C, and the gas-pressure in the main is usually 40 – 45 mm water. A number of furnaces are provided with thermotechnical control and automatic control of: (1) the proportioning of the gas and air deliveries; the pressure of the producer gas at the top of the vertical flue is used as the gas delivery pulse. Experience has shown that such a method is unreliable and this unit is not working; (2) the pressure conditions in the working space of the furnace; the pressure below the roof serves as pulse; (3) periodic reversing of the valves; it is proposed to control the temperature of the top air checkers.

The temperature of the furnace roof is measured by means of an optical pyrometer mounted on the back wall of the furnace. Of interest is a mnemonic working scheme for the furnace ends, which is included in the thermal panel of the control desk and clearly indicates the working of the right-hand and left-hand gas - air flues and stack flues of the furnace. Such a scheme could also be expediently adopted in our open-hearth steel plants in the USSR.

The following devices and methods for speeding up various operations in the melting process and improving the condition of the furnaces are deserving of attention.

* As in original – Publisher's note.

1. For improving the thermal operation of the regenerator chambers, the gas checkers are blown through regularly (once every 8 days) with compressed air at a pressure of 6 atm. gage, using ordinary $\frac{3}{4}$" gas piping.

2. The base of the gas port is made by ramming a dolomite mix or is lined with dolomite bricks, made with an addition of commercial paraffin wax (8 – 10% by weight). The dolomite used for these bricks has a particle size of up to 3 – 4 mm, and is carefully mixed with the paraffin wax, the melting point of which is 120 – 130°C. This mixture is then pressed hot under a pressure of 350 atm. gage. According to statements of the plant workers, the base of the gas port when made of dolomite prevents slag incrustation and its life is much longer than a Dinas or magnesite bottom.

3. According to information of the plant workers, at all the furnaces, molten slag is periodically removed from the slag pockets during the operation of the furnace. For this purpose, longitudinal holes 380 mm wide and 1000mm high are made in the face of the slag pockets. Normally, these holes are closed by brickwork and are opened only before running off the slag. It is, however, not possible to run off all the slag through the holes. Unfortunately, we are unable to say whether such a method of slag removal is effective, since we had no opportunity of watching the operation.

A special device, a "gun", is used for speeding up the operation of closing the steel tap-hole. This is mounted, like the charging box, on the charging machine bar. The "gun" is loaded with a moist dolomite mix which, by means of a plunger (with the movement of the charging bar), is projected into the tap-hole.

5. Before the bath is run out, the tap-hole is opened mechanically by means of an iron rod of square section 120 x 120 mm mounted on the bar of the charging machine.

The rolling plant of the works has two mills, an 850 blooming mill and a small-section mill for rolling wire, two groups of four-chamber soaking pits, each taking 16 ingots. Two three-zone furnaces are heated by hot producer gas made from brown coal. The thermal conditions of the soaking pits and furnaces are not regulated automatically. The temperature is merely measured in the furnaces by means of thermocouples in protective sheaths.

The building of a metallurgical works in Kalb was commenced in 1951 and completed in 1955. Before the plans for the works were drawn up, extensive investigations were carried out on the smelting of low-grade ores in semi-industrial low-shaft blast furnaces, using coke made from brown coal. To begin with, an experimental furnace was built with a diameter of 900 mm and a height of 1500 mm, then an experimental semi-industrial group on the Domnarvet system – a low-shaft furnace of rectangular cross-section 1500x600 mm and 3.5 meters high. The experiments on the smelting of iron ores using brown coal coke were successful and served as basis in the design of industrial low-shaft furnaces.

The first low-shaft blast furnace was started up in October 1951, and in March 1953 there were already 10 furnaces in operation. The projected output of the works, based on the production of 20 furnaces, was obtained from 10 furnaces, so that the building of further furnaces was discontinued. The mean output of a furnace is 80 tons of iron per day, the maximum being 100 tons/day. The total production of the blast-furnace plant is 800 tons daily. All the furnaces are of the same type with an effective volume of 78 m^3. The cross-section of the furnace shaft is 2300 x 7200 mm and the height 5000 mm. Two stoves are provided for each furnace for blast heating. The air heated to 800°C is supplied at a pressure of 0.33 atm. gage. Each furnace has 16 or 14 tuyeres: 7 or 6 tuyeres on each side of the long section and 1 tuyere at each end of the furnace (Fig. 7). A low-shaft blast furnace of circular section with an output of 300 t/day is at present being planned at this works.

The blast furnaces operate on poor iron ores having a high silica content. 85% of the coke of the charges is brown coal coke and 15% is bituminous coal coke. Sometimes, however, smelting is conducted on brown coal coke alone. The average consumption of burden materials is: iron ore from the Badeleb deposits 1600 kg/t, from Buchald 1000; limestone 1100, open-hearth slag 170 and coke 2000 kg/t. The average cost of 1 t of iron is 350 German marks.

The supply of stock to the furnace is mechanized. The skips containing the burden travel along a monorail; the charging apparatus is controlled from a charging desk. The stock line, temperature and blast pressure are controlled at all the furnaces.

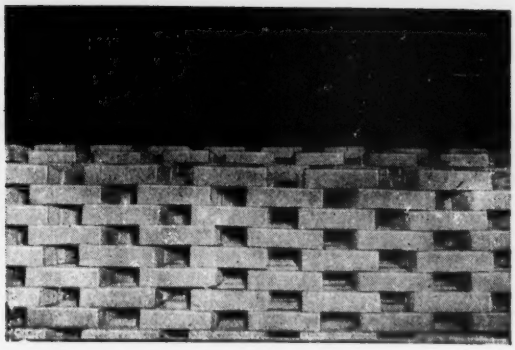


Fig. 5. General view of gas checkers

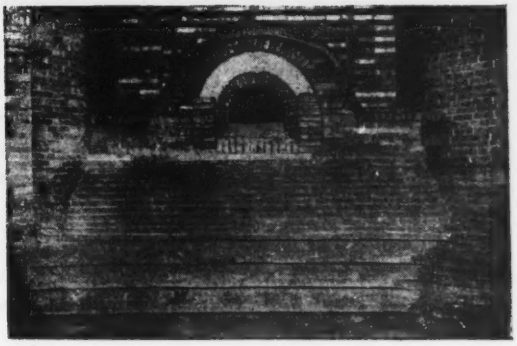


Fig. 6. Merz ports

"Max-Hutte" metallurgical works (at Saalfeld) is very old (built in 1875), but it has undergone considerable reconstruction in recent years. The works has a blast furnace plant with four furnaces (three furnaces have a capacity of 450 m^3 each and one 250 m^3).

The blast furnaces use air enriched with 25% of oxygen, the blast temperature is 600 – 750°C, pressure 0.5 atm. gage. The calorific value of the blast furnace gas is 1000 – 1100 cal/nm^3.

The Thomas shop has four 15 ton converters and two mixers, 300 and 1000 tons. There are also two 15 ton electric arc furnaces.

The converters are lined with dolomite bricks, the life of the linings is on the average 200 blows for the walls and about 40 blows for the bottom. The metal is blown for 20 – 25 minutes. Experiments are being made on one of the converters with enrichment of the air with oxygen up to 30%; the oxygen consumption is 3000 nm^3/hour. Blowing with oxygen lasts 8 mins. The blast pressure is 2.5 atm. gage.

For ordinary air blowing, 120 kg of scrap are used per ton of molten iron. The average loss of metal is 11%. The steel is top cast in ingots weighing 3 – 4 tons.

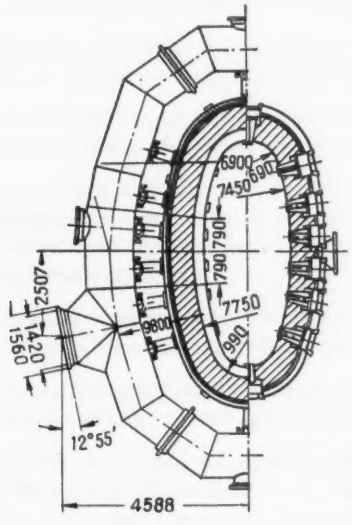


Fig. 7. Section through a low-shaft blast furnace at tuyere level.

Electric furnaces are used for melting high alloy steel, each furnace gives 3 to 4 melts in 24 hours. The roofs of the electric furnaces are lined with Dinas brick, their average life is 30-35 melts, and the life of the rammed walls (dolomite with an addition of pitch) is 60 – 70 melts.

About 90% of the entire electric steel is melted with the use of oxygen. The oxygen is blown through for 40 mins; the oxygen consumption at a pressure of 12 atm. gage is 300 nm^3/hour. The use of oxygen results in a considerable increase in the wear of the roof and walls of the electric furnaces. The works oxygen plant with an output of about 10000 nm^3/hour amply satisfies the oxygen requirements.

The rolling plant has three mills: a 1100 blooming mill, an 850 two-high mill and a 750 three-high mill. The mills are driven by steam engines. The plant has four groups of soaking pits and one continuous furnace. The construction of another continuous furnace is at present under completion. The furnaces and pits are heated by hot producer gas from brown coal and blast-furnace gas.

In the calcining plant, there is a Rhein tubular furnace 60 m long and 3.6 m in diameter. The furnace is heated by brown coal dust supplied to the plant in special tanks mounted on railroad platforms. Dust firing considerably reduces the length of service of the firebrick lining of the furnace.

In the tubular furnace iron ores containing 30% Fe and from 26 to 40% SiO_2 are calcined. Added to the charge is also waste from chemical manufacture containing 40% iron and 15% silica. Before calcining, the charge is previously ground to a particle size of less than 20 mm. As a result of the sintering process, spongy iron containing 25 – 30% Fe is obtained.

After grinding in ball mills and magnetic separation, the spongy iron contains up to 82% Fe. The average productivity of the furnace per 24 hours, according to the output of calcined product, is 300 t or 90 – 100 t of spongy iron. The agglomerating temperature of the charge is about 1180°C, and the working temperature in the tube furnace is about 1300°C.

The temperature conditions of the furnace are controlled by means of three platinum – platinum-rhodium thermocouples mounted at three points along the length of the furnace.

The enriched product with a particle size of above 1.3 mm is used in the charge for the blast furnaces and electric furnaces. The fraction below 1.3 mm is returned to the tube furnace for re-calcining. The blast-furnace burden contains 10% of spongy iron.

The visit to the metallurgical works of G. D. R. enabled us to note the following measures, the adoption of which has a definite productive effect:

(a) Comparatively extensive use of oxygen as an intensifier in steel melting and blast-furnace production.

(b) The construction of suspended system of refractory brickwork in open-hearth furnaces, providing facilitated and convenient assembly.

(c) Use of powdered iron ores in the form of briquets.

(d) Extensive use of basic slags as fertilizers in agriculture, and blast-furnace slags for cement manufacture and road ballast.

AT BRITISH BLAST-FURNACE PLANTS

G. I. Adaryukov

In September-October of 1956 a delegation of Soviet metallurgists visited a number of metallurgical works in Great Britain. The editorial department has commenced to publish a series of articles on British metallurgy with an article on blast furnaces.

During a trip to Great Britain as members of a delegation of Soviet metallurgists, we made a study of seven of the largest metallurgical works employing a complete cycle of processes (Fig. 1). The blast-furnace plants of these works smelt about 50% of the whole of the iron produced in Great Britain. In 1955, the output of iron at all the British works was 12.6 millions of tons.

The majority of British metallurgical works produce less than 1 million tons of iron annually; at the largest works, Margam-Abbey in Port Talbot, 1.5 million tons of iron are produced annually.

According to the conditions of supply of iron ore to the blast-furnace plants, British works can be divided into two groups: works in the Midlands, using local ores containing 22 – 35% Fe, and works situated near the coast using mainly imported ores with an average content of 57% Fe.

At the present time, about 60% of the total quantity of iron smelted in Great Britain is produced from ore imported from Sweden, France, North Africa and other countries.

About 75% of the coke used is produced from local coal in coke ovens forming part of the metallurgical works, the remainder is produced at individual by-product coke-oven works.

Preparation of iron ore for smelting. Together with an inspection of the blast-furnace plants, we also investigated the sections dealing with reception, storage and preparation of iron ores for smelting.

British metallurgists pay considerable attention to the organization of the transport of raw materials and fuel, bedding of the ores during storage, crushing, sorting the ores and sintering the fines.

Works using imported ore carried in ships having a displacement of 8 – 12 thousand tons possess powerful discharging equipment for rapidly unloading the ships. Thus at the Margam works there are four ore discharging cranes, each of a capacity of 500 t/hour, transferring the ore from the ships' holds to the stock yard of the blast-furnace plant.

At works receiving local ore in 30 – 35 ton cars there are stationary car dumpers. The various sorts of ore are supplied for unloading with a strict eye on the daily graph.

For the internal transport of heavy goods – coal, coke, ore and sinter – modern works use belt conveyors with a belt width of 1200 to 1600 and even 2100 mm, ensuring a large reserve of productivity.

All the ore entering the works is subjected to two-stage crushing to 50 mm in double roller and cone crushers. The Appleby-Frodingham works crush all the ore in three stages to 10 mm for subsequent sintering. The crushing and sorting departments are well organized and have a considerable reserve of power.

The ore fines of less than 10 – 12 mm are separated on vibrating screens heated by coke oven gas. This gives much better results when screening wet ores than electrical heating of the screens.

Local ores are not concentrated, since in the Northamptonshire ores, the oxides of iron and silica are chemically combined; the gangue of Frodingham ores contains 20 – 24% CaO, and it is undesirable to reduce

the basicity of the ore during concentration.

The crushed ores are sent to storage for bedding. We made a study of the work of three bedding plants of the Robins-Messiter type: for ore at the Appleby-Frodingham and Corby works, and for coking coals at the Shotton works. These plants ensure good bedding, but even with a frost of -10°C, the loading machines operate with interruptions.

Fig. 1. Location of the blast-furnace plants visited by the delegation.

The production of sinter in Great Britain was 7.8 million tons in 1955; 22 sintering machines of the belt type were in operation. The blast-furnace burden contained on the average about 30% of sinter. We inspected 16 sintering machines with a sintering area of from 45 to 94 m^2 and a belt width of from 1.83 to 2.5 m. The sinter coolers were particularly interesting: a cooler of the rotary type with 32 cars moving in a circle at the Appleby-Frodingham works, a ring cooler with forced air draught at the Shotton works and a Lurgi travelling grate cooler at the Margam works. All these coolers ensure a reduction in temperature of the sinter from 600 or 700°C to 100°C.

Sinter fluxed with lime is not produced in Great Britain, but at the Appleby-Frodingham works a mixture of ores with acid and basic gangue is sintered, giving a sinter with a basicity of 1.35, without having to introduce lime into the sinter charge.

Blast-furnace production. The planning of new blast-furnace plants at British works has a modern character and resembles the planning of blast-furnace plants in the USSR.

The plants of the largest works – Appleby-Frodingham and Margam – have four blast furnaces each, ensuring an annual output of iron of 1250 and 1500 thousand tons respectively.

Of 138 blast furnaces in Great Britain, only 41 furnaces have a hearth diameter greater than 5.5 m. We inspected 22 of the largest blast furnaces, smelting about 6.5 million tons of iron annually, with an effective volume of from 800 to 1745 m^3. The daily output of British blast furnaces is not high: furnaces giving 600 tons of iron per 24 hours are considered to be large; the productivity of the most powerful British furnaces does not exceed 600 tons of iron per 24 hours, compared with the productivity of 1800 – 2000 or more tons in the same time for furnaces of the USSR.

Figure 2 shows the profiles of three British blast furnaces. Furnace No. 1 of the Howarden Bridge works in Shotton (Fig. 2a), constructed in 1953, smelting a burden of imported ores with a 30% sinter content gives 1100 to 1200 tons per 24 hours of iron for steel manufacture. The considerable depth of dead space (1219 mm) is characteristic of recently constructed furnaces. Furnace No. 1 has 18 tuyeres.

Furnace No. 4 at the Margam works in South Wales (Fig. 2b), started up in January, 1956, is the largest blast furnace in Europe. The daily output of the furnace is 1500 – 1600 tons, with a burden of imported ores, containing 60 – 70% sinter. The proportions of the basic details of the profile of this furnaces are very similar to the proportions of the dimensions of the profiles of typical Soviet blast furnaces. Furnace No. 4 has 21 tuyeres.

The effective volume of one of the blast furnaces of the Apply-Frodingham works (Fig. 2c), constructed in 1954, was increased from 1355 to 1647 m^3 during repairs in August and September, 1956. At the present time, the diameter of the hearth of this furnace is the largest in the world (9448 mm). Smelting sinter containing 35% Fe with a basicity of 1.35 and a slag production of 1300 kg/t of iron, this furnace produces 1100 – 1200 tons of hot metal daily.

The furnace has three rows of tuyeres: a bottom row of 18 tuyeres, internal diam. 180 mm, situated as usual in the hearth, and two top rows of 9 tuyeres each, internal diameter 75 – 100 mm, arranged in staggered fashion in the upper part of the boshes.

It is considered at this works that the provision of tuyeres in the boshes enables the blast delivery to be increased and the furnace output to be raised by 10 – 15%.

A large gap (1067 – 1257 mm) between the bell and the wall of the throat is characteristic of the Appleby-Frodingham works. This is one of the reasons for the short life of the furnace linings at this works. During repairs to one of the furnaces, this gap was increased to the "record" size of 1333 mm, the top part of the throat being widened at the same time. In our opinion, this will lead to a still greater reduction in the service life of the shaft lining.

The construction of recently built blast furnaces is in line with modern blast-furnace practice.

It should be noted that lining of the well with large corrugated carbon blocks is not practiced at the present time. The use of carbon materials in the center part of the well has also been abandoned.

There is extensive use of small size carbon blocks of Carblox Ltd. (343 x 228 x 100 mm) for lining a ring 900 – 1000 mm wide in the peripheral part of the well at a depth of 3500 – 4000 mm below the center line of the iron tap hole.

The sides of the hearth as far as the center line of the slag notch are also lined with small carbon blocks, enabling the diameter of the hearth to be increased without changing the shell.

Carbon blocks are less used in the tuyere zone and boshes.

The use of carbon blocks in the shaft was stopped after the unsatisfactory results obtained in operating the furnaces at Appelby-Frodingham lined with such material.

The shaft of one of the furnaces at this works has been lined (by way of experiment) with a high alumina brick, containing 73% Al_2O_3.

The well and lower part of the hearth are cooled by vertical cooling plates or by external spraying. Extensive application is made of cooling of the tuyere zone, boshes and lower part of the shaft with horizontal box-like cooling plates.

Boshes, lined with carbon blocks, are cooled by external spraying.

The equipment of the furnace bottoms and casting beds differs little from accepted practice at blast-furnace plants of the USSR. At most of the large plants, electric guns are used. Slag stoppers are used at only a few furnaces. At the new Clay Lane works, centralized manual control of stoppers for slag and iron is employed.

The blast equipment of the furnaces is maintained in good condition, loss of blast is insignificant. At some plants, sloping tuyeres are used, with an inclination of 15°. It is considered at these plants that tuyeres of such construction burn through less frequently.

Charging of the furnaces has the following features. The burden materials are supplied to the bin gantry by belt conveyors. At new plants, the ore, sinter and lime are distributed along the bin conveyors by tipping cars, and in the older plants by transfer cars loaded by intermediate bins.

The warm climate of Great Britain makes it possible to do without rooms under the bins. The scale cars operate in the open air, considerably easing the working conditions of the scale-car operators. The load capacity of the scale cars does not exceed 20 tons.

The bins are steel, arranged in one line, with gates opened by pneumatic cylinders mounted on the scale cars.

The coke fines are sifted on vibrating screens. Grizzlies are not used. The coke is charged into the blast furnaces by weight. The charging hoists have skips of a capacity of up to 8 m^3; the skip winches have two motors.

Operation of the bells is pneumatic. We did not see electric winches for operating the bells at any of the furnaces. Usually, the large bell is secured rigidly to the rod; the opening of the bell is free. The charging apparatus enables assembly of the materials to be effected in accordance with two different programs; individual stations may be omitted in the operation of the rotary distributor.

Operating Characteristics of British Blast-Furnace

Works	Ores smelted	Sinter content of ore part of burden %	Efficiency factor	Coke consumption kg/t (skip moist)	Slag yield kg/t	Blast temperature °C	Composition of iron %				Composition of slag %			
							Si	Mn	S	P	SiO_2	Al_2O_3	CaO	MgO
Clyde Iron Co.	Imported	27—30	1.15—1.20	750—800	500—575	600—650	0.4—1.25	1.0	0.05—0.06	0.6 and 0.8—1.0	33	17	40	7
Colvilles Appleby-Frodingham	Local, poor (22-32%Fe)	96—98	1.4	850—900	1250—1300	Max. 600	0.5—0.7	1.0	0.05	1.4	34	18	39	3
Howarden Bridge	Imported	30	1.05—1.1	850	600	Max.600	0.6	1.1	0.04	0.6	30	16—18	40	10—11
Shotton	Local (28-30%Fe 0.3-0.4%S)	40—45	1.35	900—950	950—1050	700	0.9	0.7	0.21	2.05	32.5	22	34.5	7
Corby	Imported + 20 % local	60—65	1.15	770—800	600	650—730	0.4—0.5	1—1.3	0.03—0.04	0.5—0.55	30	16	41	10
Margam (blast furnace No. 4)	Imported + sinter from local ores	12—15	1.20	900	600—650	600	0.5—0.6	0.7—0.8	0.05—0.06	Morten. ≤0.8. Thomas 1.4	32	19—20	43	4

Each blast furnace has three hot blast stoves; in our view, their heating surface is inadequate in a number of cases, since it fails to ensure heating of the wind to temperatures above 650 – 700°C. Fully automatic hot blast stoves ensuring blast temperatures of up to 775°C are to be found only at No. 4 furnace of the Margam works. The equipment for automatic valve reversal was supplied by the firm of Zimmerman-Jansen; the operating mechanisms are pneumatic cylinders. At this furnace, the stack flue of the hot blast stoves is made in the form of a duct beneath the working platform of the stoves, enabling their foundations to be simplified. The method of shutting off the hot blast stoves from the hot blast main is different at different plants: by means of sliding valves or dish-shaped valves. In both cases, the life of the fittings is high, due to the use of high-grade copper castings.

For collecting the smelting products 70-ton iron ladles and slag ladles of a capacity of 10-12 m^3 are used. The slag ladles are tipped by pneumatic cylinders located on the ladle carriage.

At some blast-furnace plants (Shotton and Margam works) the slag is run into special pits situated by the side of the furnace. After cooling, the slag is loaded by excavator into self-tipping cars having a capacity of 10 – 15 tons. Such a method of slag disposal is very cheap, but can only be used where the slag is crushed for ballast and is not granulated.

At the Corby works, automobile transport is also used for the disposal of the dust from the dry dust catchers.

Data characterizing the operation of the blast furnaces inspected are given in the table.

According to the grade of ores used and the sinter content of the burden, the operating criteria of the various plants differ considerably. The furnaces of the Appleby-Frodingham and Corby works operate under the most unfavorable conditions. These furnaces operate with highest slag yield.

The Corby works employs sulfide ores with an acid gangue, producing iron containing 0.18 – 0.25% S. The iron is treated with a soda mixture in the ladle after tapping the iron and after the mixer, with a total soda

consumption of 10 – 12.5 kg/t of iron, the sulfur content being reduced to 0.045 – 0.055%.

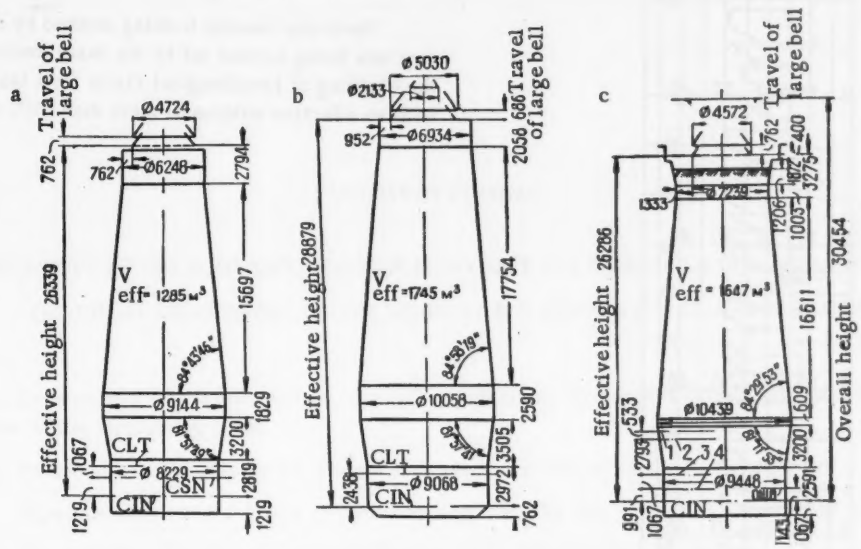


Fig. 2. Profiles of three British blast furnaces: CLT – center line of tuyeres; CSN – center line of slag notches; CIN – center line of iron tap-holes; 1) center line of the 9 tuyeres of the third row; 2) center line of the 9 tuyeres of the second row; 3) center line of the 18 tuyeres of the first row; 4) center line of first and second slag notches; 5) center line of third slag notch.

At the Ebbw Vale works, part of the iron sent to the basic steel plant is treated with soda mixture. Other British works use soda for desulfurizing iron only in isolated cases.

In view of the relatively low intensity of smelting (650 – 800 kg of coke burnt in 24 hours for 1 m^3 of effective volume of the furnace), the blast furnace operation is unsatisfactory from the point of view of our practice. At the new large furnaces, forced checks are made systematically (Fig. 3) – up to 70 or 80 times in 24 hours.

The blast-furnacemen of the Appleby-Frodingham works explain the necessity for such a practice as due to the special working conditions: high ratio of volume of sinter to volume of coke and high slag yield.

It should be mentioned, however, that even at the new blast furnace of the Margam works, smelting a rich burden with a slag yield of 0.6, the charge is checked every 20 mins. and this is explained as being based on the experience of the blast-furnace plant of the Appleby-Frodingham works.

British blast-furnacemen do not control the working of the gas current and distribution of the materials in the furnace: the number of measuring instruments provided is small, the gas is not sampled over the radius of the throat. Mention should be made of the poor durability of the shaft linings at the Appleby-Frodingham works and the systematic growth in the central part of the shaft at the Corby works.

The advanced technological working methods adopted in our plants find little application in Great Britain. Only three blast furnaces (No. 2 at the Clyde Iron works and furnaces at the Bilston and Consett works) operate with increased top pressure, the gas pressure being maintained at a level of only 0.4 atm. gage. In view of this, the results obtained from changing furnaces over to increased top pressure are not very significant.

Likewise, blast of constant humidity is not used, due to a considerable extent to the inadequate capacity of the hot blast stoves.

Mention should be made of the considerable interest displayed by British metallurgists in Soviet metallurgical practice.

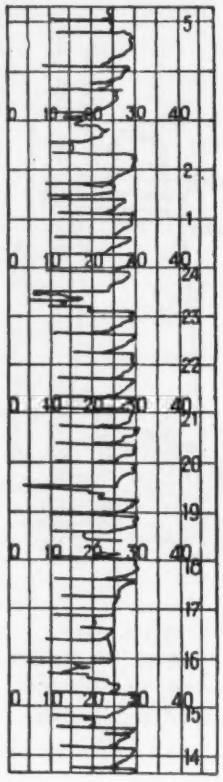


Fig. 3.

Our technical journals are studied and partly translated into English.

Particular interest is being aroused by the project now being worked out by the State Institute for the Planning of Metallurgical Plants for a blast furnace with an effective volume of more than 2000 m^3.

COMPETITIONS

ANNOUNCED BY THE SCIENTIFIC-TECHNICAL SOCIETY FOR FERROUS METALLURGY AND THE TECHNICAL DEPARTMENT OF THE MINISTRY FOR FERROUS METALLURGY OF USSR

1. Development of the construction of a high-temperature blast-stove for blast furnaces, including the construction of the air-ducting plant.

2. Development of a scheme for the complex automatic control of a blast furnace.

3. Development of new methods of removing slag from the slag pockets of open-hearth furnaces.

4. Development of methods of clearing deposits from open-hearth regenerators.

5. Improved constructions of machines for trimming, edging, straightening and sorting rolled products.

Any person, as well as any undertaking, institute and scientific research and planning organization may participate in the competitions.

The following prizes will be awarded for the best suggestion in each competition:

First prize - one - 15000 roubles

Second prizes - two - 7500 roubles each.

Incentive prizes of from 1000 to 500 roubles each will be awarded for suggestions failing to satisfy all the conditions of the competition, but suitable for introduction into industry.

The last date for submitting suggestions for the competition is 30th June, 1957.

Suggestions with the superscript "For the competition of the Scientific-Technical Society for Ferrous Metallurgy" should be addressed to the Central Office of the Society at: Moscow, G-34, 2nd Obydensky periulok, No. 14, Room 13, under an arbitrary motto (name and address should be given separately in a sealed envelope).

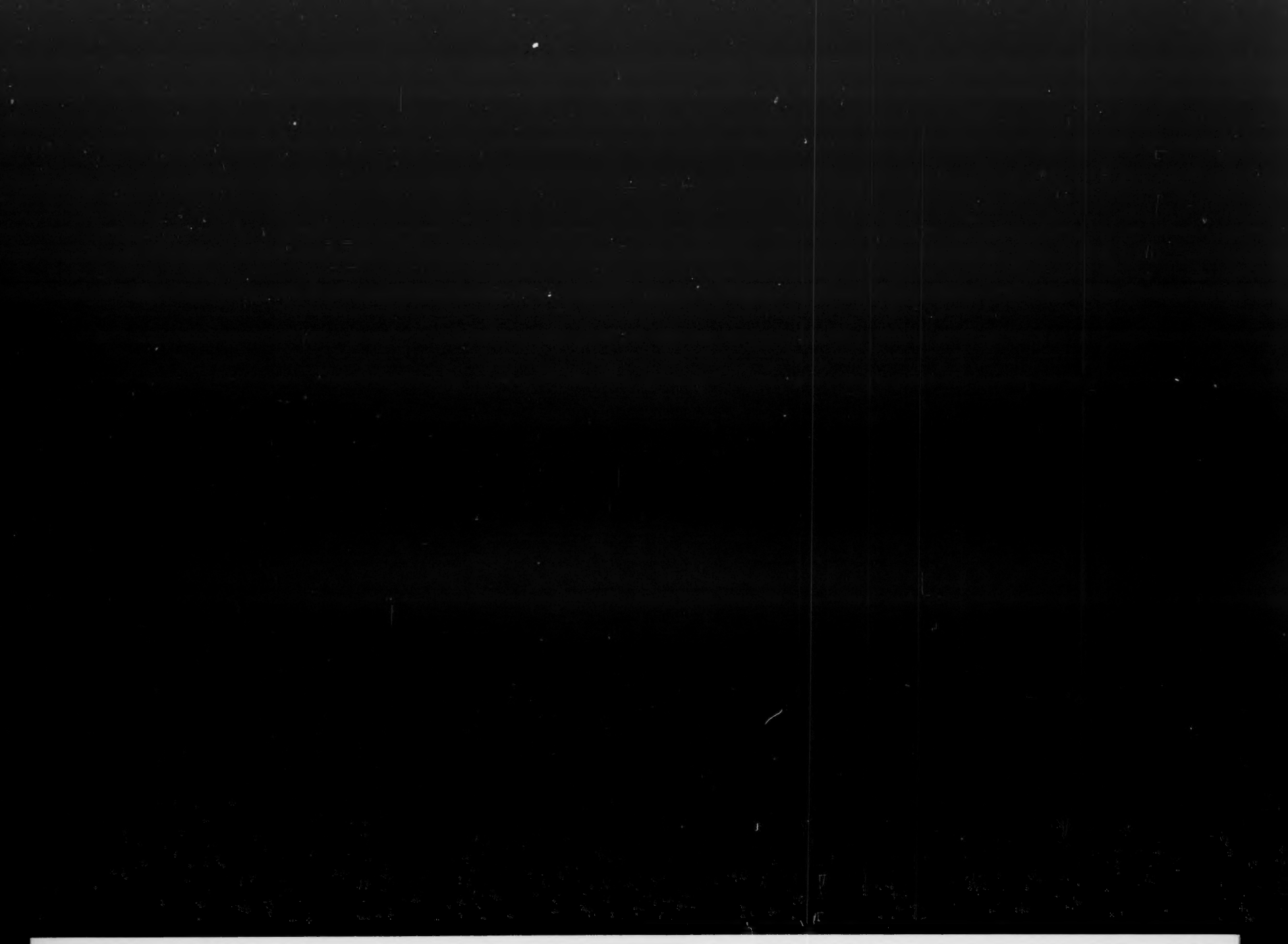
The competition conditions can be obtained from the principal organizations of the Scientific-Technical Society for Ferrous Metallurgy in ferrous-metallurgical undertakings and institutes, as well as from the central, republic, district and inter-district offices of the Society.

METALLURGIST IN ENGLISH TRANSLATION

March, 1957

TABLE OF CONTENTS

	Page	Russ. Page
1. Better Quality Production, Less Spoilage. I. P. Zubrev	139	1
Projects of the Sixth Five-Year Plan		
2. The Sokolov-Sarbai Ore Concentration Plant. N. F. Sandrigailo	141	3
Blast-Furnace Production		
3. Continuous Temperature Control in the Combustion Zone of a Blast Furnace. I. F. Koval and A. Ya. Kravchenko	145	6
4. Measures for Speeding-Up Blast Furnace Overhauls. V. N. Mashyanov	149	8
5. Iron and Steel in 1956.	152	10
Steel Smelting		
6. The Smelting of Structural Chrome Steel in 380-Ton Open-Hearth Furnaces. B. G. Petukhov	153	11
7. Protection of Radiation Pyrometers from Damage by the Furnace Flame. V. M. Studentsov	158	14
8. Iron and Steel Ministry USSR	162	16
Rolled Steel and Tube Production		
9. Influence of the Slab Rolling System in the Blooming Mill on the Quality of the Metal Surface. L. V. Andreyuk	163	17
10. Experience in the Better Utilization of Rolls. S. P. Orel and N. U. Paukov	168	20
11. Improved Operation of Sheet Mill Rolls on the Roughing Stand. S. F. Kumachev and I. D. Dmitriev	170	21
12. Lengthening the Take-Up Roll of a Blooming Mill. S. G. Dorman and V. I. Khvatov	172	22
13. Centrifugal Casting of 50 mm Cast Iron Standpipes. B. D. Khakhalin and A. N. Smolyakov	176	25
14. Screening and Ventilation of Rolling Mill Control Cabins. Ya. A. Shtromberg and A. F. Babalov	179	27
Experience of Innovators		
15. Working Methods of Sintering-Machine Foremen A. M. Buziner and M. S. Petrenko. V. Lovchinovsky	182	29
Experience of New Techniques		
16. Organization of Roll Changing at Mill 650. L. I. Boldyrev and K. S. Gudovshchikov.	185	31

TABLE OF CONTENTS (continued)

	Page	Russ. Page
New Books		
17. Iron Producers Criticize Scientist's Book	189	33
Schools of Advanced Experience		
18. Advanced Experience in the Organization of Blast-Furnace Repairs. D. I. Felman	190	34
New Techniques and Rationalization		
19. New Type of Door Frame for Open-Hearth Furnaces. G. A. Sokolov	196	38
20. New Construction of Air-Cooled Charging Door. S. M. Andonyev, O. V. Pilipyev, V. F. Volkov and B. M. Sirenko	199	40
Metallurgy Abroad		
21. At the Conference of German Engineer-Metallurgists. A. K. Adabashyan	201	41
22. At British Blast-Furnace Plants. G. I. Adaryukov	207	45
23. Competitions	213	Inside back cover